

Production of Ferroalloys

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INTRODUCTION

The Soviet ferroalloy industry is a post-Revolution development. Its firstling, the Chelyabinsk Ferroalloy Smelter, was commissioned in 1931 and it was not long before the U.S.S.R. stopped importing ferroalloys. The Chelyabinsk Smelter was followed by the Zestafon, Zaporozhye, Aktyubinsk and other ferroalloy smelters.

At present, the Soviet ferroalloy industry possesses up-to-date smelters producing all necessary ferroalloys.

Mastery of modern techniques and furnace technology requires thorough theoretical knowledge and practical know-how. This, in turn, is achieved by a profound study of the theory and practice of ferroalloy production methods and continued improvement of workers' skill.

Great attention is now concentrated on the high-quality steels needed for the development of the chemical, aviation, electrical and other industries.

It is planned to increase considerably the output of alloy steels, whose properties are enhanced by alloying elements, such as chrome, nickel, manganese, tungsten, molybdenum, niobium, titanium, vanadium, etc. These elements are sometimes introduced into steel in pure form; the usual practice, however, is to add them in the form of ferroalloys, for their production is much simpler and cheaper. Moreover, it is considerably easier to add most of the elements to steel when they are in the form of ferroalloys. For example, metals such as tungsten and molybdenum are characterised by high specific gravities and elevated melting points and are slowly assimilated by a steel bath. The iron alloys of these metals possess lower specific gravities and lower melting points, and that makes it much easier and more convenient to use them in the manufacture of steel.

Steel may be alloyed simultaneously by several elements. Steels are classified according to the alloying element assay as low-alloy, medium-alloy and high-alloy steels.

According to their use, steels are divided into three main groups: constructional, tool, special and alloy.

Constructional steel is used for the manufacture of machine parts, motor-cars, aeroplanes, building elements. It can be carbon or alloy steel. The mechanical properties of alloy steel are considerably

higher than those of carbon steel. Chromium and nickel are the main alloying elements used for constructional steel.

Tool steel is used for the manufacture of tools (lathe knives, chisels, cutters, gauges, etc.); it is either carbon (with 0.7-1.2% C), or chrome-, manganese-, silicon- or tungsten-bearing alloy steel.

High-speed steels, alloyed with tungsten, chrome and vanadium, belong to the tool-steel class; they acquire considerable hardness after the heat treatment and retain it at high temperatures (500-600°C).

Special-quality steels include corrosion-resisting, stainless, acid-resisting steels and heat-resisting steels and alloys capable of bearing considerable loads at high temperatures.

Alloys with 75-78% Ni and 20-23% Cr possess high ohmic resistance and are used for the manufacture of furnace resistance elements. The combination of several alloying elements produces steels and alloys with a wide range of properties.

Ferroalloys make it possible to solve another very important problem: degassing and deoxidising of steel. Oxygen and other gases dissolving in steel during the smelting process evolve when the metal cools, causing pipes and gas cavities. The gases dissolved in steel weaken its mechanical properties. To eliminate these gases it is necessary to effect a technological operation called deoxidation. This is accomplished by the addition of the ferroalloys of silicon, manganese and titanium, which combine with oxygen to form stable oxides that do not dissolve in steel. These elements are called deoxidisers.

Titanium and zirconium, combining with nitrogen to form stable insoluble compounds, are added to steel to eliminate yet another gas—nitrogen.

Alloying elements in the form of ferroalloys are also used in the manufacture of high-quality pig iron.

Part I

FERROALLOY FURNACES

Ferroalloy furnaces are divided into two main groups:

- 1) ore-smelting furnaces,
- 2) refining furnaces.

The former are used for smelting all grades of ferrosilicon, most of the manganese alloys, as well as the high-carbon grades of ferrochrome. Refining furnaces are used for smelting refined grades of ferrochrome, ferromanganese and several other alloys.

Ore-smelting ferroalloy furnaces are very much similar in design to electric furnaces used for smelting pig iron, copper-nickel matte, calcium carbide, etc. These furnaces usually have transformers of more than 7,000 kW.

Ore-smelting furnaces may be of one- or three-phase type. In the Soviet Union one-phase furnaces are built for special purposes only and for very limited use. The bath of a one-phase furnace is cylindrical in shape with a conductive carbon hearth. The hearth is embedded with copper bus-bars. The transformer is connected to the hearth and to the electrode by a flexible cable. The one-phase Miguet furnace (Fig. 1), which has a power factor of 0.95 despite extremely heavy current (250,000 A), is a splendid specimen of efficient design.

Charging is done by machines and the charge loosened by pneumatic pokers. On the whole, however, the Miguet furnaces are expensive and complicated and, consequently, are not widely used.

There are also one-phase two-electrode furnaces but they too are not used on a large scale.

Three-Phase Ore-Smelting Furnaces

In three-phase furnaces the electrodes are arranged either in a line (rectangular furnaces) or at the vertices of a triangle (circular furnaces). Large furnaces may have as many as six electrodes.

Rectangular furnaces (Fig. 2) have the following drawbacks:

- 1) each electrode forms independent reaction crucible; this necessitates operation of three separate tapholes;

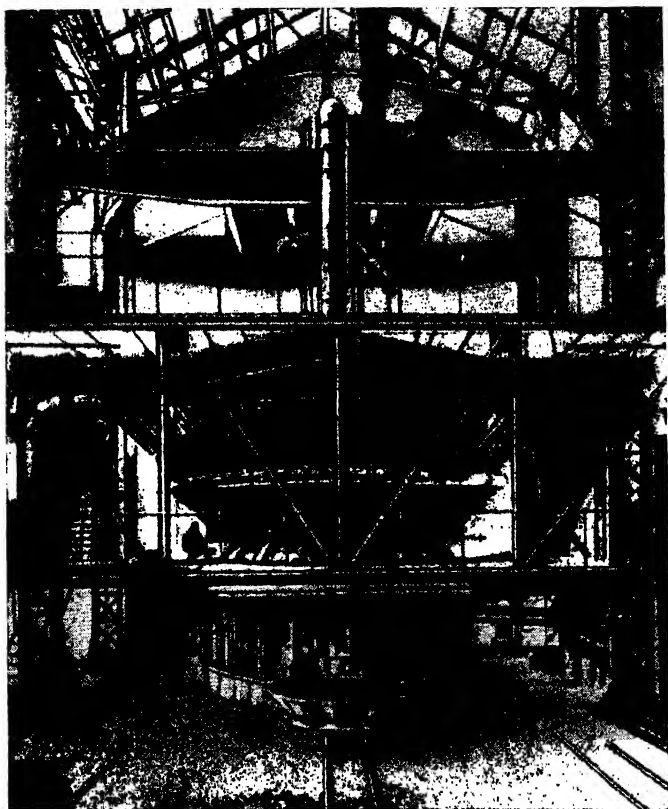


Fig. 1. One-phase Miguet electric arc furnace of 10,000 kVA

2) the power factor is low because of great distance between the outer electrodes;

3) "wild" (unstable) or "dead" (glow discharge) effects are sharply expressed.

Three-phase circular furnaces are used most in the ferroalloy industry. Fig. 3 shows the cross-section of the transformer and furnace aisles of 10,000-kVA three-phase circular furnace.

In a circular furnace with electrodes at the vertices of a triangle the melting crucibles are connected with each other, making it possible to operate the furnace with one taphole.

If the low-voltage circuit is well designed, these furnaces may have a high power factor, reaching 0.95-0.96 in the case of powerful furnaces, and minimum "wild" and "dead" phase effects.

Furnaces with rotating baths. As soon as the charge reaches the high-temperature zone (particularly in the process of smelting silicon alloys) it melts down and sinters. The fines obtaining in the charge fill the space between the bigger particles of the charge, sharply reducing the latter's gas permeability, creating bridges and thus hampering the descent of the charge as well as the escape of gas. To restore normal operating conditions the charge is moved by poles and metallic *pokers*. Moreover, the reaction crucible frequently contracts as a result of technological disturbances. Much effort has to be spent to broaden it. To eliminate this trouble a new type of rotating bath furnace has been designed (Fig. 4).

Smelting alloys in a rotating bath furnace improves process technology because it ensures effective gas permeability, eliminates accretions at the furnace top, facilitates the work of operators, prolongs the life of furnace lining, since the high-temperature zone moves constantly with respect to the hearth and the sides of the furnace.

The speed of rotation of the bath is calculated by the following formula:

$$t = \frac{2\pi^2 R_c y d h}{N \alpha},$$

where t — the time necessary for the complete rotation (360°) of the furnace in hours;

R_c — the radius of the electrode circle, in centimetres (with the electrodes situated at the vertices of an equilateral triangle);

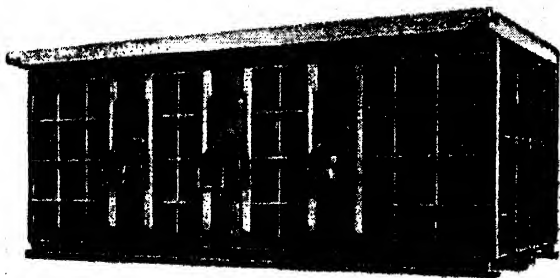


Fig. 2. Shell of a rectangular 7,500-kVA furnace for smelting ferromanganese

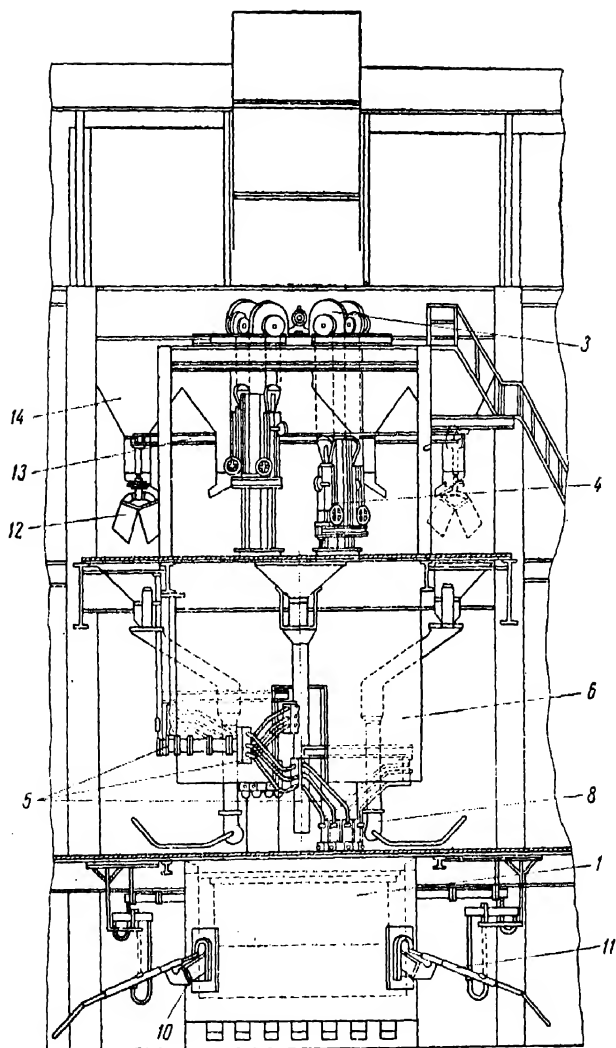
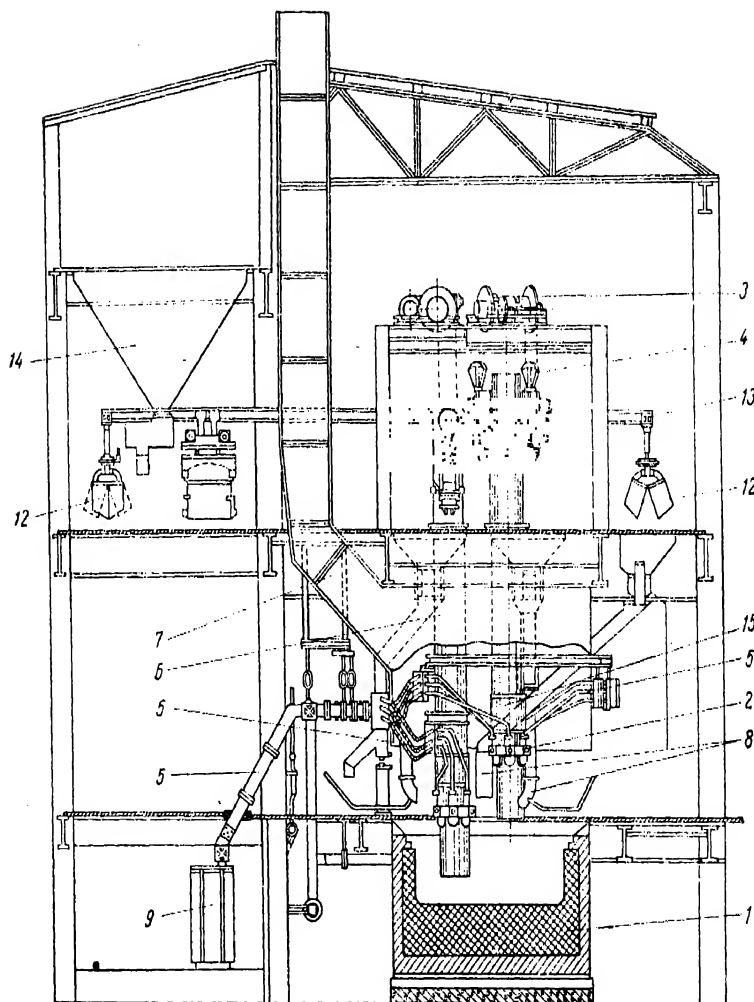


Fig. 3. Cross-sections of a transformer and furnace aisles of a shop
 1—lining carbon blocks; 2—electrode-holder ring; 3—electrode-positioning
 former and the furnace; 6—exhaust hood; 7—gas-evacuating flue; 8—charging
 ing car; 13—monocrall; 14—mixture-



equipped with circular three-phase furnaces of 10,000 kVA:

mechanism; 4—electrode suspension; 5—low-voltage circuit between the trans-
pipes; 9—furnace transformer; 10—taphole; 11—tapping apparatus; 12—charg-
storing bin; 15—electrode mast

γ —weight of the charge per unit volume, t/m^3 ;
 d —diameter of electrodes, cm;
 h —depth of immersion of the electrodes, cm;
 N —power input per electrode, kW;
 α —weight of the charge melted and reduced per 1 kWh, ton.

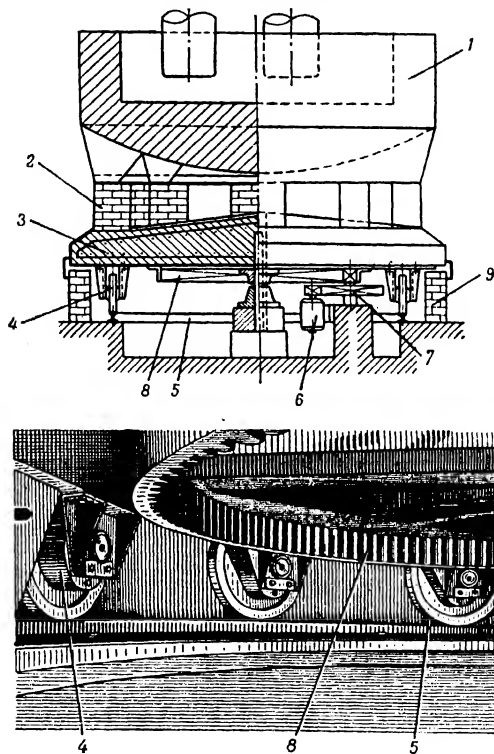


Fig. 4. Rotating furnace for smelting ferrosilicon:

1—furnace hearth; 2—stools; 3—concrete slab; 4—rollers;
 5—circular rail; 6—motor; 7—reduction gear; 8—crown gear;
 9—wall

Optimum rotating speed, however, is determined by practice.
 In the U.S.S.R. furnaces of this type are operating successfully
 at the Chelyabinsk Ferroalloy Smelter. The yield per man is 20%
 above that of the stationary furnaces due to higher furnace output
 and economy of furnace-top operating time.

Closed-top furnaces. Ferroalloy furnace gas consists mainly of carbon monoxide and has a calorific power of $2,500 \text{ kcal/m}^3$. Closed-top furnaces are currently used in the ferroalloy industry to recover escaping gases.

In design, closed-top furnaces (Fig. 5) are similar to those with an open top, except for the dome-shaped roof.

The water-cooled dome has a non-magnetic framework and is lined with refractory concrete or with specially shaped firebrick (Fig. 6). The roof is provided with safety valves as a precaution against excessive gas pressure in the furnace (Fig. 7). Materials are charged into the furnace hoppers by pipes (Fig. 8); a certain amount of mixture is always left over in these hoppers; it serves as a shut-off valve. Surplus pressure in the furnace dispenses with air leakage.

The furnace is equipped with a flue and a dust precipitation system.

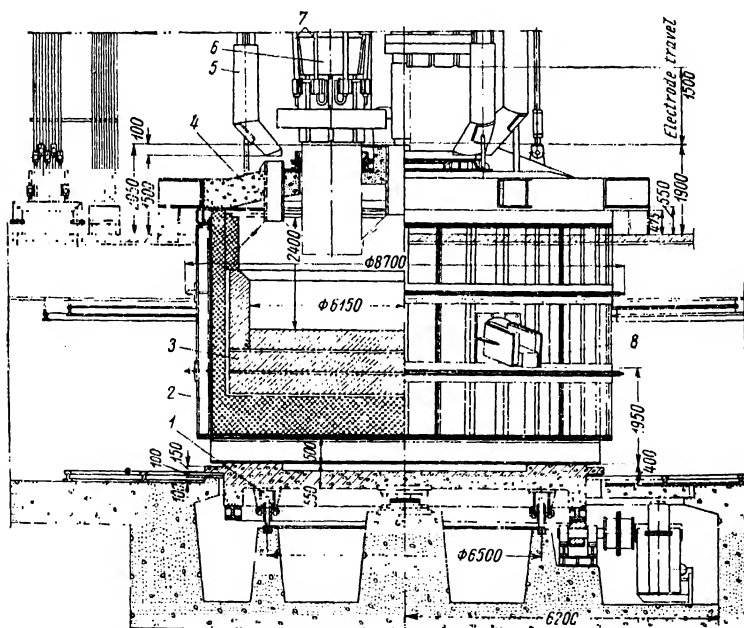


Fig. 5. Closed-top 16,500-kVA rotating furnace for smelting ferrosilicon:

- 1—rotating mechanism; 2—furnace shell; 3—hearth lining; 4—roof; 5—charging device; 6—electrode holder; 7—water-cooling system; 8—taphole

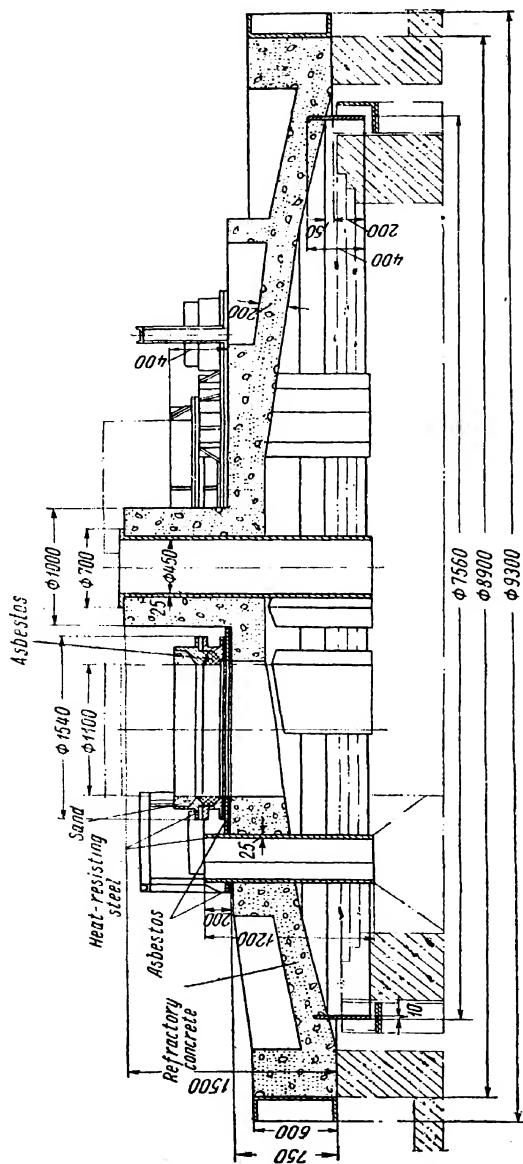


Fig. 6. Castable roof of a closed-top rotating furnace

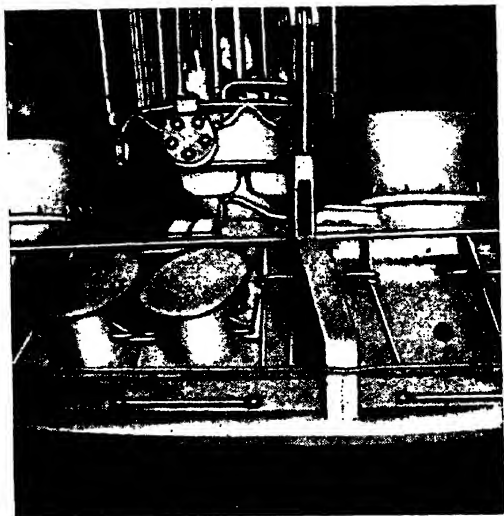


Fig. 7. Roof of a closed-top ore-smelting furnace with safety valves and hoppers for mixture charging

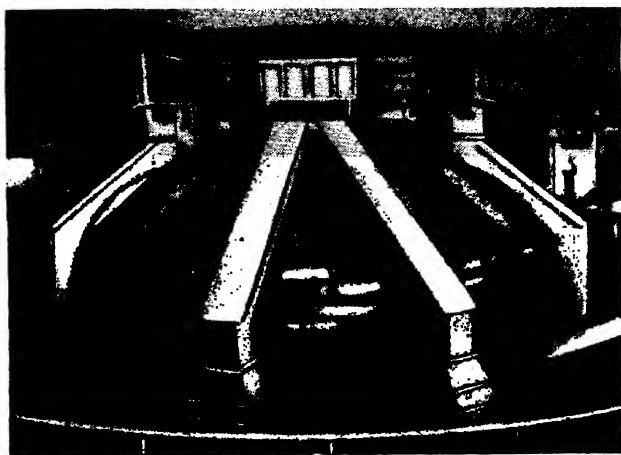


Fig. 8. Charging pipes, hoppers and power supply circuit of a closed-top furnace

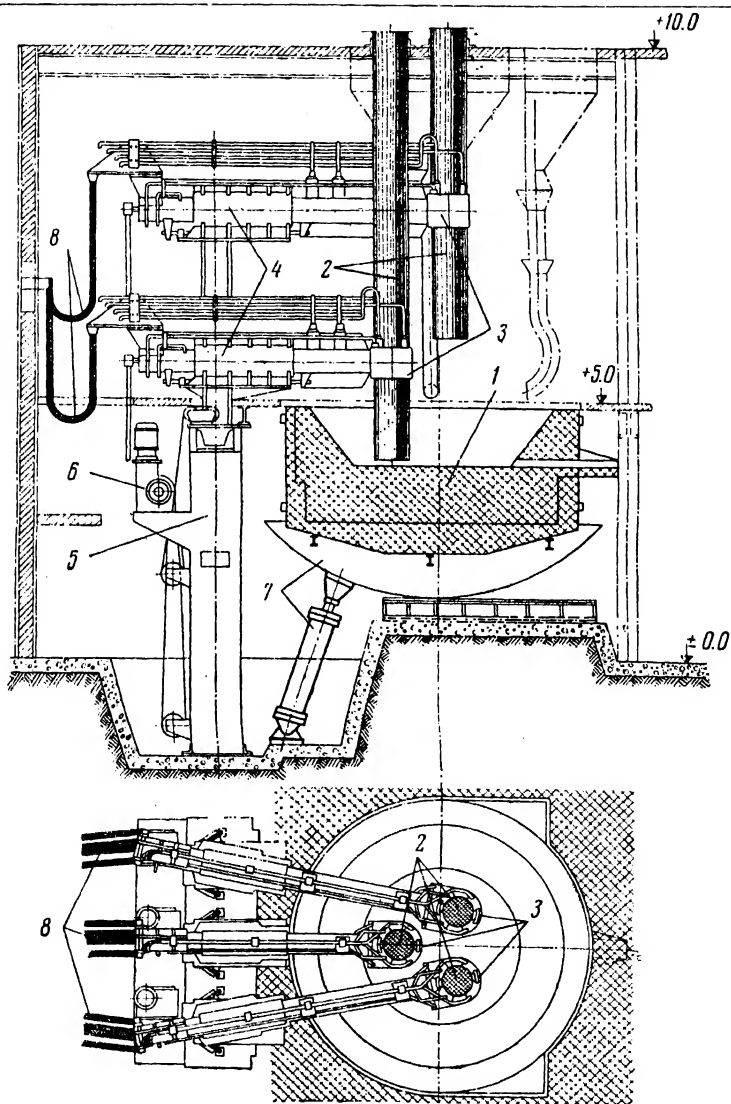


Fig. 9. Ferroalloy rotating-bath furnace for refining processes:
 1—hearth; 2—electrodes; 3—electrode holders; 4—mobile columns; 5—stationary
 columns; 6—electrode-positioning motors; 7—furnace-tilting mechanism; 8—low-
 voltage circuit

Refining Furnaces

Refining furnaces are stationary or tilting.

Tilting or rocking furnaces (Fig. 9) have an advantage over the stationary ones as far as refining processes are concerned. They permit quick tapping, slagging, servicing and lining repairs.

On the other hand, their maintenance requires greater skill and their manufacture and installation bigger outlays.

As for their design, only the bath (Fig. 9) or the whole furnace structure with the electrodes (Fig. 10) may be tilted.

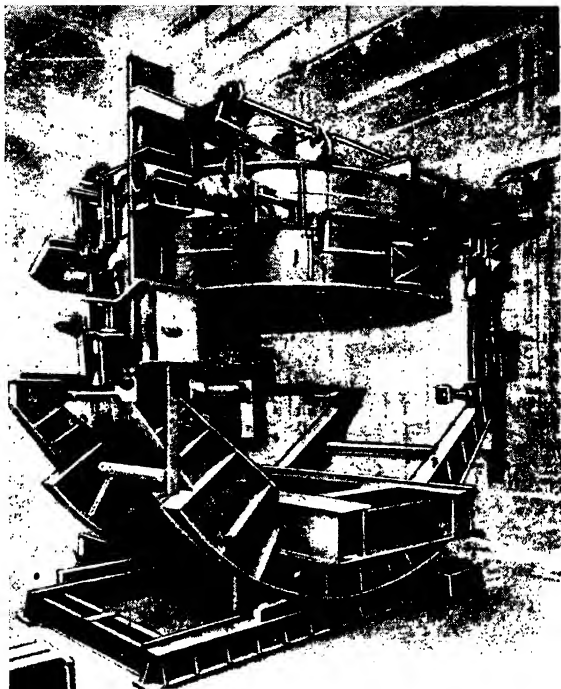


Fig. 10. Steel framework of a modern three-phase refining furnace

Fig. 11 shows a refining furnace with a rocking and rotating bath used for smelting ferromanganese.

New types of standard ferroalloy furnaces are now being designed in the Soviet Union with due account of operating practice and technology requirements.

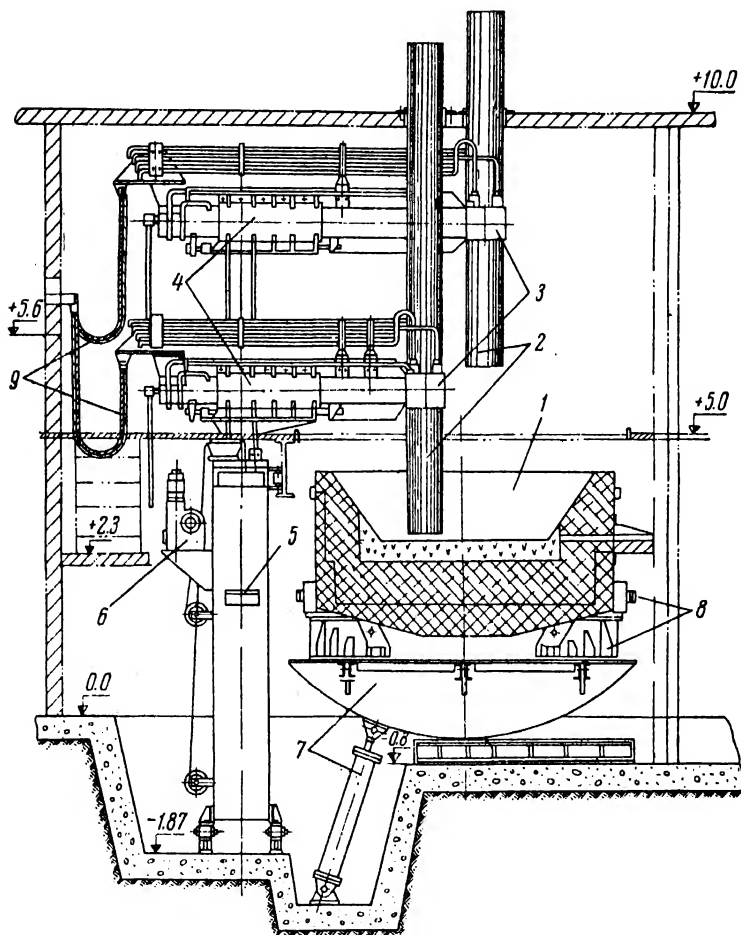


Fig. 11. Ferroalloy furnace of 2,500 kVA with rotating and tilting hearth for smelting refined manganese:

8—rotating mechanism; 9—low-voltage circuit (the rest see Fig. 9)

Chapter 1

FERROALLOY FURNACE HEARTH DESIGN

Main Structural Features of Ferroalloy Furnace Hearths

The shape of the hearth of a ferroalloy furnace depends on the arrangement of electrodes. One- or three-phase furnaces with equilateral-triangle arrangement of electrodes have a circular hearth; if the electrodes are arranged in a line, the hearth is rectangular or elliptic. The dimensions of the hearth are determined by the designation of the furnace and its rating.

The main design features of a ferroalloy furnace hearth are the following:

- 1) electrode diameter and diameter of the electrode circle;
- 2) diameter and depth of the hearth;
- 3) diameter and height of the furnace shell.

The chief design characteristics of the ferroalloy furnaces now in use are given in Table 1. The table also provides data on specific power consumption parameters which relate hearth dimensions to furnace power rating.

The main characteristics of hearth performance are its campaign and heat losses through the side walls and the bottom. Furnaces smelting silicon alloys are usually relined after seven to ten years, while the campaign of furnaces smelting refined ferrochrome lasts only one year. Heat losses range from 0.5 to 3% of total power consumption.

Furnace Shell

The shell of a ferroalloy furnace should be strong enough to withstand all the stresses imposed at elevated temperatures by the weight of the lining, the raw materials and the metal, as well as the pressure exerted by the furnace lining as the result of heat expansion when the furnace is fired.

Shells of stationary furnaces are manufactured from 15 to 25 mm steel sheets either welded together or riveted. When carbon lining is used, the seams should be hermetically welded to prevent the lining from burning out.

Table 1

Main Design Characteristics of Ferroalloy Furnaces

Rating, kVA	Type of furnace	Alloys smelted	Electrode diameter, mm	Diameter of electrode circle, mm	Hearth dimensions		Shell dimensions		Specific power rating, kVA		
					diameter, mm	depth, mm	diameter, mm	height, mm	per 1 m ² of electrode hearth area	per 1 m ² of circle area	per 1 m ³ of hearth vol.
46,500	Three-phase closed-top with rotating hearth	Silicon and carbon	1,200	2,900	6,450	2,400	7,500	4,575	558	2,496	232
46,500	Three-phase open stationary	"	1,200	2,900	6,300	2,400	7,500	4,300	529	2,496	221
43,500	Ditto	"	1,100	2,700	5,800	2,400	7,300	4,300	511	2,353	213
40,500	Ditto	"	1,000	2,500	5,300	2,300	6,800	4,200	476	2,138	207
7,500	Ditto	"	900	2,300	4,500	2,300	6,000	4,200	472	1,807	205
3,500	Three-phase open-top with rocking and rotating hearth	Refined ferrochrome	450	1,200	3,000	1,400	4,300	—	495	3,067	354
2,500	Ditto	Refined ferrovanadium	500	1,200	3,000	1,200	4,300	—	354	2,212	295
2,500	Three-phase open-top with rotating hearth	Ferrotungsten	500	1,200	2,800	1,200	4,000	—	406	2,212	338

To make the design more rugged standard channels are welded on the inside along the generating lines of the cylinder; moreover, three or four horizontal stiffening hoopings, made of sheet metal and rolled stock or of rolled stock alone, are welded onto the shell.

In the taphole area the shell is reinforced either by a steel casting or by sheet metal, which carries cast or welded taphole fittings.

Large ore-smelting furnaces with circular hearths are provided with one or two tapholes. In the case of two tapholes, one is in operation and the other in reserve; the latter is tapped periodically to keep it in operational condition. If technology requires separate tapping of metal and slag, two tapholes are provided: one for metal and the other for slag; if metal is separated from slag outside the furnace, the second taphole is not necessary. When electrodes are positioned in line, individual tapholes are arranged opposite each electrode.

Some Soviet smelters operate today with magnesite lined furnaces without shells; four stiffening belts made of horizontally and vertically positioned channels are provided. Tapholes are made of welded steel plates and fastened to the columns which brace the stiffening hoopings. The furnace bottom rests on I-beams or channels, as in furnaces with shells.

The rotating-hearth furnace shell is so mounted that its bottom, fastened to parallel I-beams (which also serve as air-cooling ducts), rests on a reinforced-concrete plate. The running wheels of the reinforced-concrete plate rest upon a circular rail embedded in the

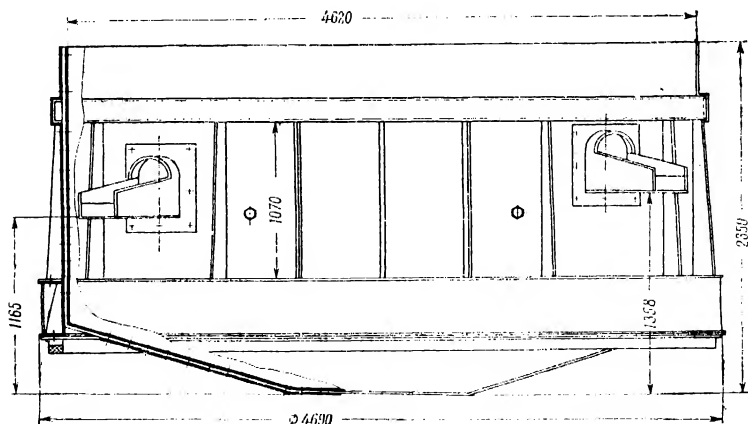


Fig. 12. Shell of a 3,500-kVA rocking furnace

furnace foundation; the rail is strictly trued up both horizontally and vertically.

The shell of a rocking furnace should be extra rugged and strong because it undergoes considerable stresses when the furnace is tilted. The shell bottom in this case is shaped like a trunked cone (Fig. 12).

A hooping in the form of a tray is fastened to the upper edge of the shell in closed-top furnaces and serves as a stiffener as well as a seal (sand or liquid) to prevent gas evolution.

Refractories

The life span of refractory lining has a serious effect on the furnace performance: output, costs, etc.

Furnace burnouts caused by low-quality refractories or poor lining lead to considerable down-time and large losses of electric power for furnace heating after repairs.

Refractories in furnace linings are exposed to thermal, chemical and mechanical wear and should, therefore, possess high physico-chemical properties.

A metallurgist should have a perfect knowledge of these properties and should be able to determine the quality of the refractories, and ensure adequate maintenance after properly lining a furnace.

Refractories in ferroalloy production are used not only for furnace linings but also for casting alloys and for lining roasters and crucibles for thermic metallurgy.

The following most important properties should be provided in refractories:

Refractoriness is a capacity of a material to withstand high temperatures. It is characterised by the temperature at which a standard specimen of the material loses its initial form by gravity.

Fire resistivity is a capacity of a material to retain its initial form under load at high temperatures. It is characterised by the temperature at the beginning of deformation under a load of 2 kg/cm².

Spalling resistivity is a capacity of a material to retain its mechanical strength under repeated sudden changes of temperature. To determine spalling resistivity six normal bricks are heated up to 850°C and then cooled in running water. These procedures are repeated until they lose 20% of their initial weight, the number of consecutive temperature changes being the characteristic of spalling resistivity.

Chemical stability or slag stability is a capacity of a material to resist, at high temperatures, chemical corrosion by slags or melts, which form easy-fusing compounds with refractories. To test the

refractory, it is submerged in melted slag heated to a given temperature.

Mechanical strength is determined by the capacity of a material to resist compressive stress. It should be noted that the strength of various refractories varies with the rise of the temperature. For example, at normal temperatures, a magnesite brick is 2 to 3 times as strong as a silica brick, while at 1600°C a silica brick is 2 to 3 times as strong as a magnesite brick. Heated up, some refractories tend to contract, others to expand. Contraction leads to a loss of structural strength, while expansion may entail the crushing of brickwork.

Heat conductivity is a capacity of a material to conduct heat; the heat conductivity of most of the refractories rises with the temperature. Lower heat conductivity of lining materials means lower heat losses and electric power consumption per ton of alloy.

Electric conductivity is characterised by specific resistance.

Refractories are acid, basic or neutral, depending on their chemical composition.

Silica brick is an acid refractory. The raw materials for its manufacture are quartz and quartzite. These materials are almost pure SiO_2 . Prepared silica contains 92-96% SiO_2 , the rest being impurities (Al_2O_3 , Fe_2O_3 and CaO). CaO is added mainly as a binder and mineralising agent; to this end from 2 to 2.2% of lime water is added to the silica mixture providing mineralisation. Silica-softening temperature (1650°C) is close to its fusion point (about 1750°C). Silica is seldom used in ferroalloy production. The use of silica in ferroalloy furnaces may be recommended only for coating carbon linings in smelting silicon alloys. This is done to protect carbon lining during the heating-up periods, as well as for heat insulation (particularly lightweight silica).

Crushed magnesite is a basic refractory, manufactured by calcining natural magnesite at 1650°C. Calcining at this high temperature prevents subsequent moisture absorption and contraction. Crushed magnesite is used for sintering furnace hearths (in low-carbon ferromanganese smelting practice), for filling the magnesite lining joints of furnaces smelting refined ferrochrome, and for filling the space between the furnace shell and its lining. This filling serves as thermal insulation and bumper which takes all the stresses caused by thermal expansion.

Magnesite brick is manufactured from crushed magnesite. Iron scales are added to the mixture to improve calcination, and 2-2.5% of finely ground red clay (alumina) and approximately 5% of water are added to provide better heat resistivity. Bricks pressed from this mixture are fired at about 1650°C. The fired bricks consist of periclase crystals cemented by silicates. Magnesite bricks assay at

least 90 % MgO and at most 3 % of CaO. The specific weight of magnesite brick is 3.5-3.6, its refractoriness about 2000°C.

Magnesite bricks well withstand basic slag effect but lack fire resistivity: sudden temperature changes cause spalling. The electric conductivity of magnesite bricks rises with the temperature. Loss of mechanical strength comes at temperatures considerably below its fusion point, since its softening temperature is 1500°C.

Magnesite bricks are used widely in the hearth and wall lining of furnaces smelting ferrochrome, ferrotungsten and low-carbon ferromanganese.

Smelted magnesite is obtained by smelting crushed magnesite in an arc furnace. The smelting of crushed magnesite removes impurities such as Fe_2O_3 and Al_2O_3 and raises the MgO content to 98%. The temperature for softening smelted magnesite is 1675°C.

Practice has shown that resistivity of furnace linings with smelted magnesite is higher than that of usual magnesite; so is its cost.

Firebrick—a neutral refractory—is manufactured from fired refractory clay and contains approximately: 62 % SiO_2 , 35 % Al_2O_3 + TiO_2 , 2 % Fe_2O_3 , 0.7 % CaO and 0.3 % MgO. The presence of ferrous oxide and calcium oxide is undesirable because it considerably lowers the fusion point of a firebrick. The refractoriness of a firebrick is 1670-1730°C, the softening temperature—1150-1350°C, specific weight—2.5-2.7 g/cm³. Its heat resistance is high, and heat conductivity and electric conductivity are low.

The firebrick is the cheapest of refractories and is used wherever technically possible. It is used for the thermal insulation of ferroalloy furnace hearths and walls as well as for the lining of all furnace elements not exposed to high temperatures. Ladles and launders may also be lined with it.

Carbon blocks—carbon refractories—are able to withstand temperatures up to 2000°C. They are made of anthracite or fine coke, with pitch as a binder.

Carbon blocks are distinguished by high stability against acid as well as basic slags containing no oxides, capable of being restored by carbon at given temperatures. The heat-resistivity of carbon blocks is satisfactory; rapid cooling causes cracks. High mechanical strength is a feature of the refractory at both low and high temperatures. Its heat and electric conductivity is high. At temperatures above 500°C carbon linings should be protected against air or gases containing oxygen, as well as against water vapour since carbon oxidises at these temperatures. Consequently, furnace shells lined with carbon blocks should be airtight. Adequate servicing prolongs the service life of a carbon lining to five or more years.

Carbon blocks are widely used for lining furnaces smelting silicon alloys and ferrotungsten, as well as for lining tapholes and launders.

Lining of Ferroalloy Furnaces

The choice of refractories for furnace lining is determined by the technology of smelting, by the character of the slag and alloy composition. The ferroalloy furnace lining is distinguished by the

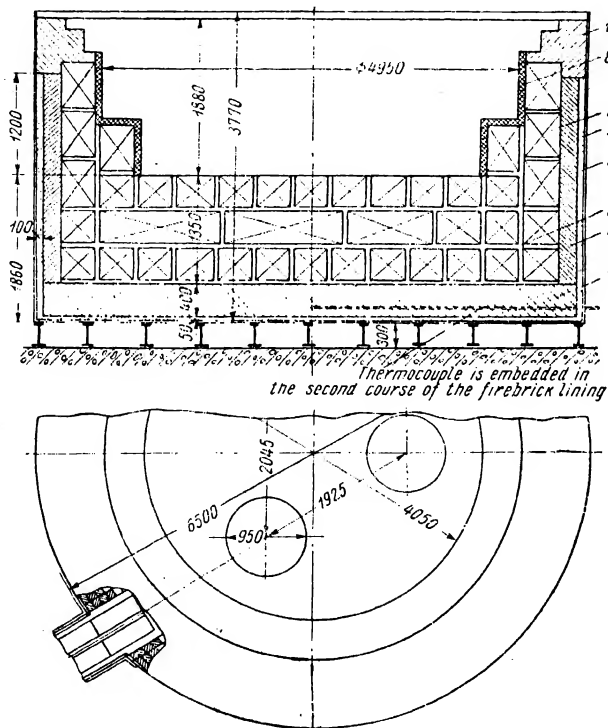


Fig. 13. Lining of a 11,500-kVA ferroalloy furnace for smelting silicon alloys:

1—firebrick lining of walls; 2—carbon blocks; 3—firebrick filling; 4—sheet asbestos; 5—anode-paste tamping; 6—firebrick lining; 7—firebrick insulation layer; 8—silica brick

considerable thickness of the hearth which ensures greater thermal inertia and stability of temperature in the smelting zone of a furnace during short idling periods. Further, a fact common to all ferroalloy furnaces is that the actual working layer of the furnace lining is the

so-called coating, i.e., accretions formed by burden components, lining fragments and smelted alloys.

One method of lining a large ore-smelting furnace for silicon alloys is shown in Fig. 13. The hearth bottom is first lined with sheet asbestos, then a 50 mm layer of crushed fireclay, and this is followed by an insulating layer comprising six courses on the flat of the firebrick; the second and third courses are masoned with mortar, while the rest are dry masonry. A thermocouple is embedded in the second course of the brickwork in the centre of the furnace. A layer of ramming mixture (anode paste with a lesser amount of pitch) is tamped over the firebrick. Three courses of carbon blocks ($1,400 \times 400 \times 400$ mm) are laid over the latter; such a lining presents a minimum amount of joints. Block joints are tamped with a ramming mixture by means of special metallic tampers which are heated before being used to provide for a more compact filling of joints. The breadth of a joint is 70 mm, since a narrower joint cannot be tamped satisfactorily. The furnace shell is lined on the inside with asbestos sheet which is topped by an insulating layer of three on-edge courses of firebrick.

A crushed fireclay filling approximately 100 mm thick is provided between the insulating layer and the asbestos. This filling serves as a precaution against shell deformation and rupture due to the expansion of the lining during the heating-up periods. Channels left in the lining in addition to this provide for heat expansion as well as for the evacuation of gases when the furnace is heated up. These channels are subsequently filled with crushed fireclay. Hearth walls, 550 to 800 mm thick, are lined with carbon blocks to a height of 1,300 mm from the bottom. The walls above the carbon blocks are made of firebrick (laid on edge with a crushed fireclay filling).

The inside furnace lining is faced with brick, preferably silica, to preserve carbon blocks from oxidation during the heating-up process.

The taphole is lined as follows: a carbon block approximately 2 m long is inserted into the upper course of the bottom lining; its external end reaches out into the taphole opening of the furnace shell. The remaining cavities between the fittings, blocks and lining are filled with anode paste. The furnace shell is cut out above the taphole block and the so-called arch is tamped with anode paste. This tamping is periodically renovated, approximately every five to seven days, by means of a special iron flap behind which the finely ground mixture is tamped. The mixture is sintered due to the heat from the furnace. Another method of taphole lining is shown in Fig. 14.

Liquid metal runs from the taphole along the block into a ladle or the receiving end of a runner and then through a launder into a mould. The receiving end and the runner are made either of sheet

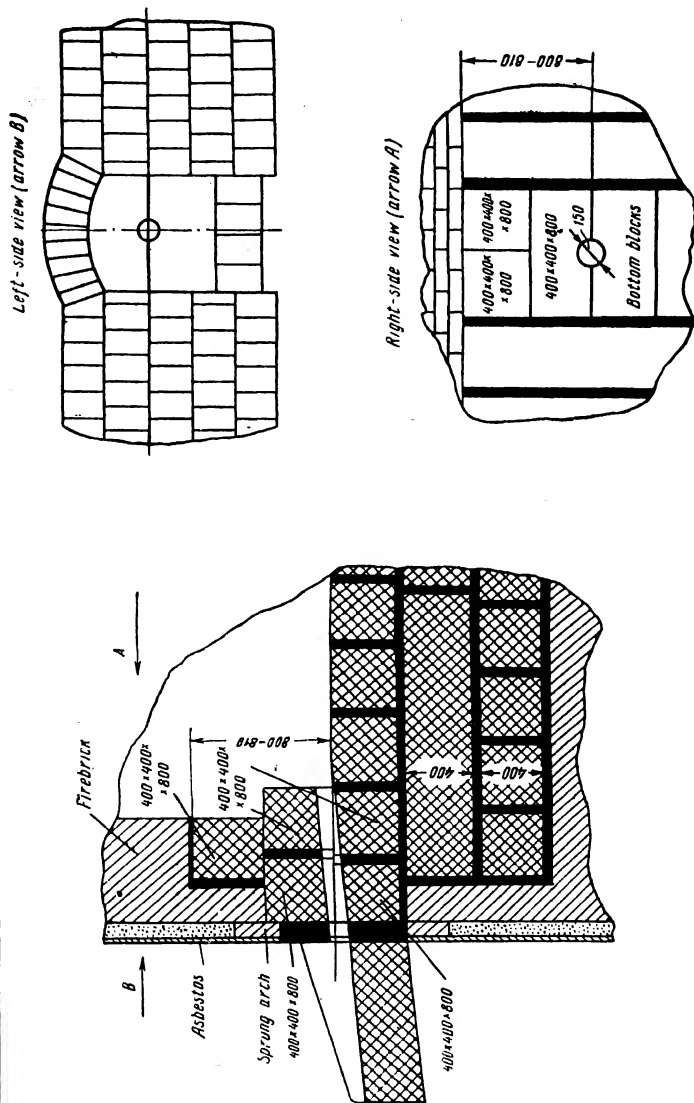


Fig. 14. Taphole lining of a 7,500-kVA ferroalloy furnace for smelting silicon alloys

metal or sections of iron castings. In the first instance, the runner is lined with carbon blocks, in the second with firebrick. In both cases the lining is covered with tamped anode paste.

Furnaces for smelting ferrochrome, refined ferromanganese, etc.,

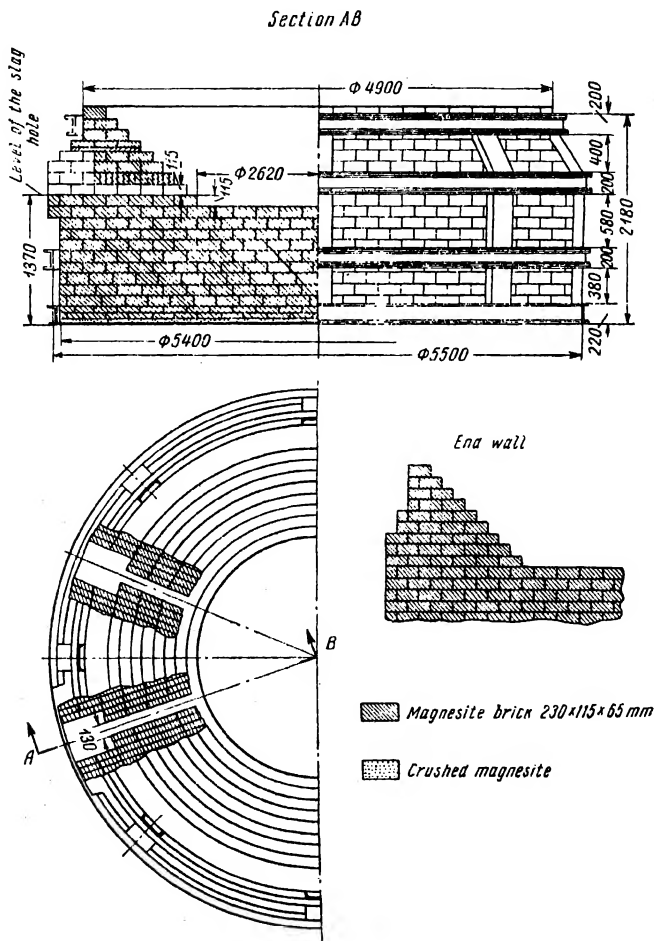


Fig. 15. Lining of a 3,500-kVA ferroalloy furnace for smelting refined ferrochrome

are lined with magnesite brick. One type of furnace lining for refined ferrochrome smelting is shown in Fig. 15. The furnace bottom is lined with sheet asbestos and then covered by a 50 to 80-mm layer of fireclay powder. This is followed by an insulating course of firebrick on the flat and the magnesite bottom (from six to eight courses of dry-on-edge masonry). The joints of each course are filled with magnesite powder (crushed magnesite passed through a 2-mm sieve).

Two types of brickwork are used: herringbone and bonded brickwork. Herringbone brickwork is mostly used, for the bonded pattern requires bricks of absolutely the same dimensions for each course; their selection is a labour-consuming operation.

The walls are lined with magnesite brick 1,600 mm thick at hearth and 230 mm thick at platform level. The use of wedge bricks is advisable for better lining performance.

The furnace is provided with two tapholes: one for slag and the other for the alloy. The eye of the alloy taphole is 150-200 mm above the hearth level. The level of slag hole is determined by the maximum alloy level prior to tapping; usually the slag hole is 150 mm above the alloy taphole. Launderers (both for slag and alloy) are lined with magnesite brick.

The furnace campaign depends to a great extent on the quality of furnace lining.

Drying and Heating Operations

Drying and heating operations are important procedures which have a direct effect on the lining campaign.

The duration of drying and heating procedures is determined by the time necessary to attain normal productivity and normal specific power consumption per ton of alloy.

The sequence of drying and heating a furnace (from 8,000 to 12,500 kVA rating) as well as of the electrode baking procedure is as follows.

1. Prior to drying, the electrode hoisting mechanism is so adjusted that in its lowest position the electrodes are approximately 100 mm from the hearth and the level of anode paste in the furnace shell is 2 m above the clamps.

2. The drying of the hearth and electrode baking are started with firewood. Drying with firewood should last at least 16 hours.

3. Fine coke is added following the drying with firewood; its amount should be sufficient to form a pad 200-250 mm thick on the hearth. Electric power is then applied.

4. During the initial four hours of operation the electrode current is kept at approximately 7,000 A. Power is switched on for one hour and off for the next.

5. The drying process is regulated so as to attain $\frac{2}{3}$ of nominal power input at the termination of 48 hours from the moment power is first applied. The furnace operates until $\frac{2}{3}$ of the rated power

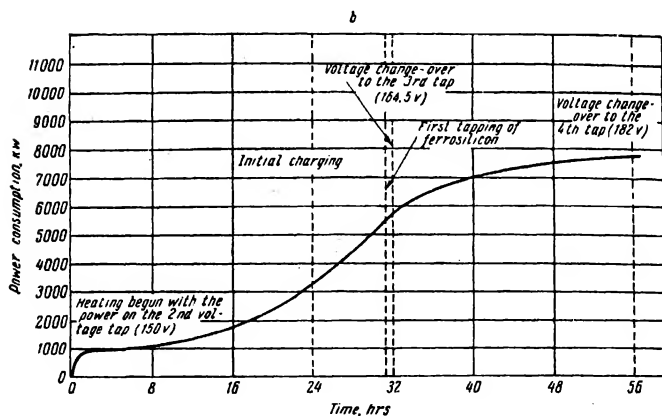
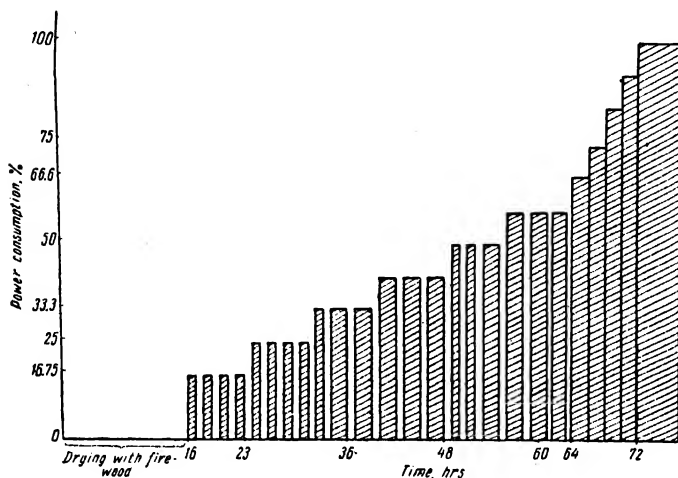


Fig. 16. Approximate heating-up schedule for a furnace smelting silicon alloys

input level is reached with idling intervals of minimum 30 minutes every hour without charging the mixture.

6. The burden is charged when power consumption attains $2/3$ of rated power consumption. The amount of burden charged prior to the opening of the taphole should average two-three charges per hour.

This minimum charge is necessary to protect furnace equipment from overfire; charging at a more frequent rate than the one mentioned above may cause the furnace bottom to freeze, and this lowers furnace efficiency.

7. The taphole is opened when from 20,000 to 25,000 kWh of electric power have been consumed from the moment of initial charging.

8. Raw materials are charged at a normal rate after the first tapping and the attainment of rated power input.

The furnace is considered heated when the temperature of the bottom reaches approximately 600°C (sample heating time-tables are given in Fig. 16, *a* and *b*).

Prior to heating a refining furnace intended for smelting refined ferrochrome, its bottom and side walls are fettled with a mixture of ore and lime (1,800 kg of ore and 1,650 kg of lime). The furnace heating procedure follows approximately the time schedule given below.

1. The furnace is charged with materials (comprising 1,800 kg of ore, 1,650 kg of lime and 360 kg of ferrochrome silicon) which are then melted; furnace power is switched off every 60 minutes to level off lining temperatures.

2. When the first charge has been smelted a second batch, consisting of 1,800 kg of ore, 1,650 kg of lime and 360 kg of silicochrome, is charged and smelted as in the first instance.

3. When the second batch has been smelted a burden with a normal weight of ferrochrome silicon (1,800 kg of ore, 1,650 kg of lime, 720 kg of ferrochrome silicon) is charged. Its smelting requires 3,000 kWh. It is smelted much in the same manner as the two preceding batches. Slag is tapped when the smelting has been completed.

4. After the third charge has been smelted two normal charges are melted and slag is tapped into a ladle; when the third charge has been melted the taphole for the metal is opened and both the metal and the slag are tapped.

The heating procedure of the above-mentioned type of furnace requires about 50,000 kWh of electric power.

The furnace is considered to be heated when the temperature of the bottom reaches 400°C .

Chapter 2

MECHANICAL EQUIPMENT OF FERROALLOY FURNACES

The mechanical equipment of a ferroalloy furnace comprises the following mechanisms:

- 1) electrode holder, its suspension, slipping and positioning mechanisms and electrode winch;
- 2) furnace-cooling system;
- 3) burden-conveying and charging systems;
- 4) fan ventilation;
- 5) electrical tapping apparatus.

Electrode Holder, Its Suspension, Slipping and Positioning Mechanisms

Two types of electrode holders are used in ferroalloy furnaces: freely suspended and rigidly connected to a mobile carriage or arm.

Furnaces with a Freely Suspended Electrode Holder

Furnaces equipped with suspension-type electrode holders are widely employed: such electrode holders are installed on all large ore-smelting furnaces as well as on some refining furnaces.

The most important assembly unit in an electrode holder is the clamp. It is meant to hold the electrode in a definite position and to ensure perfect electrical contact between the clamp and the electrode.

The electrode clamp operates in high-temperature conditions and consequently should be provided with a reliable water-cooling system (Fig. 17); this system should be sufficiently rigid and, whenever possible, made of heat-resisting materials. Until recently ring clamps consisting of a ring, clamping screws with nuts and contact clamps, were used.

Contact clamps are cast of copper or copper alloys—bronze or tombac; these alloys are distinguished by low resistivity and satisfactory heat conduction, which makes for their reliable functioning. Satisfactory operating results are obtained with contact

clamps made of chromous copper. Clamps may be hollow or with a cast-in steel (preferably copper) tube for water cooling.

Contact clamps are rectangular (for a 12,000-kVA furnace clamps dimensions are $1,000 \times 333 \times 60$ mm) with a rounded lower edge. Clamps made of copper water-cooled piping have shown good operating results.

Current density in the clamps usually averages $1.4\text{--}1.7$ A/cm². To prevent current leakage between the clamp and the ring (that is very important from the point of view of operational reliability) the clamp and the pressure screws are insulated with asbestos or micanite which are inserted into a special clamp housing at the point where the clamp contacts the clamping screws or the thrust block. Insulation is protected by a steel plating inserted between it and the screw.

Current to the contact clamps is supplied by copper pipes 70 mm in diameter. Clamps are suspended from the electrode shell by insulated lashings which ensure the flow of electric current only from the clamp to the electrode.

The ring is composed of two halves (Fig. 18) interconnected by two steel pins through bronze sleeves. The use of half-rings precludes a continuous magnetic circuit which is formed by current flowing through the electrode. It is advisable to manufacture rings out of non-magnetic heat-resisting alloys which eliminate electric power losses on ring magnetising, as well as the need for water cooling.

Half-rings are hollow welded or cast steel casings with pockets containing specially shaped clamping nuts. The number of clamping screws corresponds to that of the clamps (six to eight in large furnaces).

Nuts are made of cast iron or, better still, of bronze. The recess on the outside of the ring follows the shape of the nut. The nut inserted into the recess is rotated 90°, then immobilised by two stop pins; the latter prevent it from rotating, while the flange of the recess prevents it from falling out. The pins are consolidated by a plate inserted into guides welded to the upper surface of the ring. The nut is inserted and extracted together with the screw as it is often impossible, in emergencies, to unscrew the nut quickly.

Clamping screws are loosened on the outside of the ring (two or three screws) when it is necessary to slip the electrode. The screws are loosened and tightened without laying the furnace down. Wrenches should be well insulated while workers slipping the electrodes should position themselves on non-conducting supports.

A number of electrode holders equipped with remote-control slipping mechanisms have been designed to facilitate the slipping of electrodes as the work with long and unwieldy socket wrenches

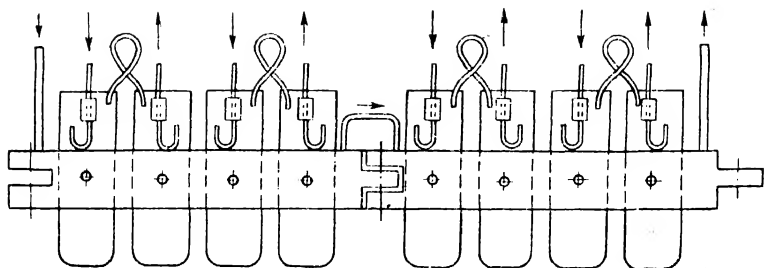


Fig. 17. Diagram of the water-cooling system of an eight-clamp electrode holder

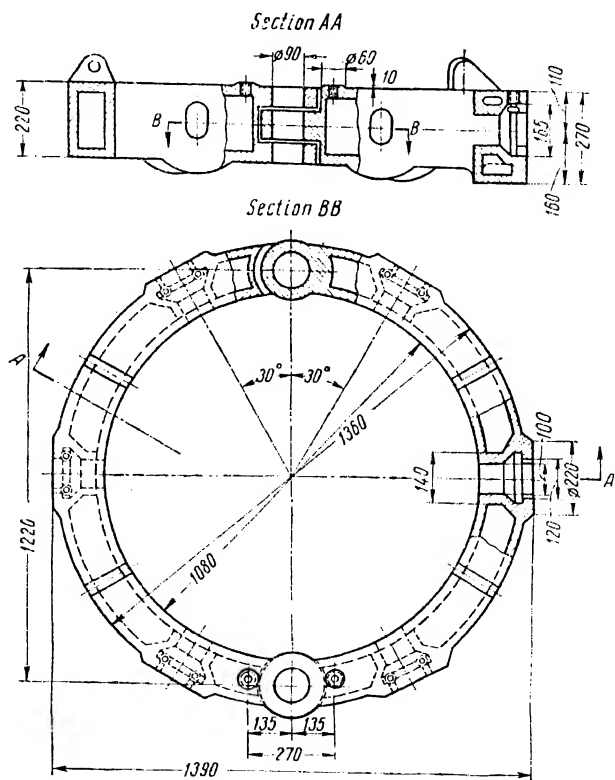


Fig. 18. Electrode-holder ring for a 750-mm electrode

is exhausting. A hydraulic clamp is used in some designs (Fig. 19). Pressure in the system in this case equals 25-30 kg/cm²; the clamping device drives the clamp close against the electrodes. When an electrode is to be slipped pressure in the system should be removed, and the clamping device then releases the clamp. A furnace equipped with such a device is shown in Fig. 20.

A number of designs of spring-type clamping devices have been recently worked out in the U.S.S.R. In this device the pressure on the clamp exerted by two springs is adjusted by a regulating screw. A somewhat simpler clamping device of the same type has been successfully tested and is now used at the Chelyabinsk Ferroalloy Smelter (Fig 21). Electrode slipping in a furnace equipped with a spring clamping device requires only repositioning the limiting blocks and loosening the friction grip of the braking mechanism. This greatly reduces the time and labour necessary for the electrode slipping operation.

Practice has shown that spring clamps are absolutely reliable. Spring clamps permit electrode slipping without power input limitations; a perfect "clamp-electrode" contact is ensured on the clamps thanks to even and constant spring pressure; this prolongs the service life of the clamps and drastically reduces their burnout failures. Spring clamps are at present installed in large Soviet-made furnaces.

In a freely suspended electrode holder the weight of the electrode acts on the electrode shell (made of 8-10 mm sheet steel) which envelops the entire length of the electrode.

The electrode shell diameter is 100-150 mm larger than that of the electrode. The clearance between the shell and the electrode serves as a channel for air delivered into the cylinder by a blower. The air is directed down to the clamps. The upper end of the electrode shell is provided with an asbestos or rubber gasket; fan air is fed to a wind box welded onto the gasket through a goffered hose. A 10,000-kVA furnace is equipped with a 40,000 m³/hr blower.

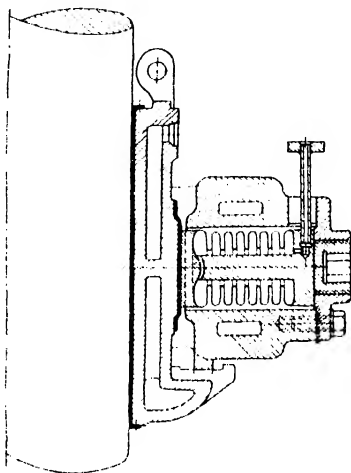


Fig. 19. Arrangement of the clamping device

A furnace cannot be operated long without the electrodes being air-cooled, which makes it necessary to install a spare blower for a group of furnaces. Individual blowers installed on the upper end of the electrode shell may also be used for the air cooling of electrodes. Thus the electrode shell solves simultaneously a number of problems: suspension of electrodes, their protection from heat radiation and furnace top gases, cooling of electrodes for anode paste conditioning. The air fanning of the electrodes protects their surface above the clamps from dust, thus ensuring reliable contact between the clamp and the electrode.

It should be pointed out that in winter warm air sometimes has to be fed into the shell as electrode-baking temperature conditions are disturbed by cold air and the electrode paste is insufficiently heated. The result is that the electrode-baking process becomes abnormal, reducing the mechanical strength of the baked electrode and worsening electrode duty because of higher oxidising reactivity.

In certain cases a hollow water-cooled box (shields of the electrode shell) consisting of four to eight sections made of 3 to 4 mm sheet steel, is fastened onto the lower end of the electrode shell to improve electrode performance. The clamps and the ring of the electrode holder are fastened through insulated supports to a ring provided at the lower end of the electrode shell. Water-cooled cross-bars carrying mobile slipper blocks and copper water-cooled conducting tubes are secured to the centre of the electrode shell.



Fig. 20. Furnace of 7,500 kVA for smelting ferromanganese, with 1,100-mm electrodes and clamping drums

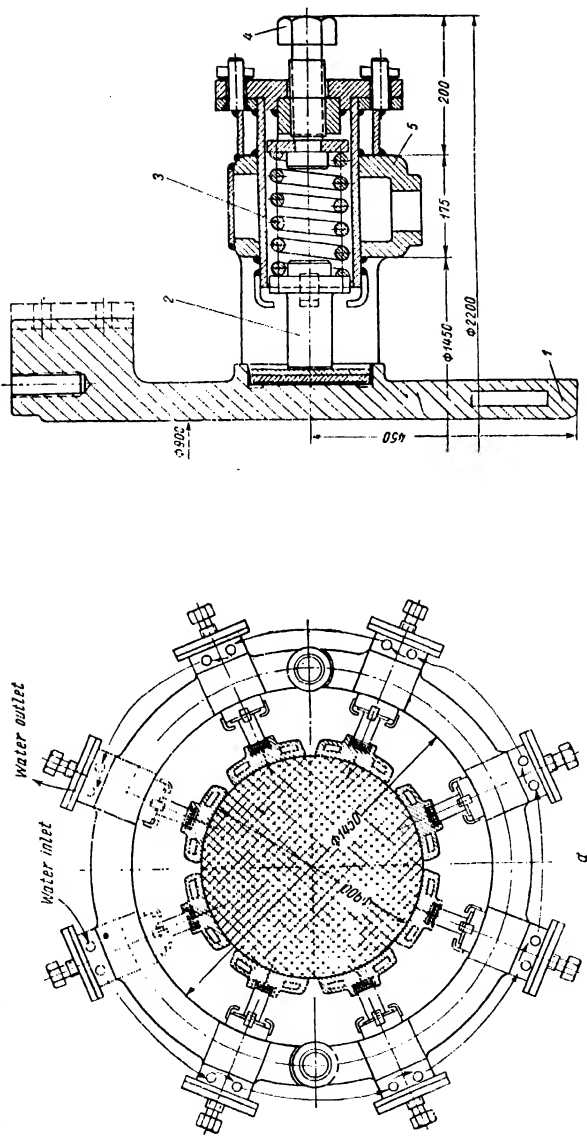


Fig. 21. Spring clamping device, designed by Chelyabinsk Ferroalloy Smelter for a 900-mm electrode:
a—general view; b—spring clamp; 1—contact clamp; 2—spring; 3—thrust; 4—regulating screw; 5—electrode-holder ring

In suspension-type electrode holders, the framework, made of channels and possessing two vertical uprights, is welded to the upper end of the electrode shell. The uprights are topped by a pulley casing. Steel cables are reeved in the pulleys and wound on the winch drum of the hoisting mechanism located on the electrode platform. One end of each cable is fixed to the winch frame, the other is secured

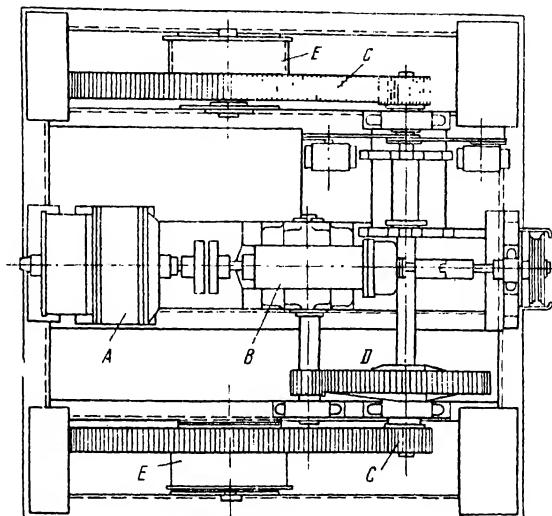


Fig. 22. Electrode-positioning winch

to the winch drum. The drum is driven by a motor and a self-locking transmission gear. The pulley casing is insulated from the uprights, as the uprights, the frame and the shell are usually under voltage.

The mechanism for feeding the electrode shell is shown in Fig. 22. It consists of motor *A*, worm transmission gear *B*, two pairs of spur pinions *C* and *D*, and two drums *E*.

Rating of motors varies with the presence or absence of the counterweight. Made of massive iron plates it balances the weight of the electrode and the electrode holder. The speed of electrode travel is within 0.3 to 0.6 m/min. Each winch is provided with a safety catch to limit hoisting or lowering.

Compact gear winches or hydraulic devices are being extensively used of late as drives for electrode-feeding mechanisms.

Electrode-Slipping Devices for Furnaces with Freely Suspended Electrode Holders

The electrode should be slipped as it is consumed, i.e., its working end should be elongated. The electrode is slipped by means of a special braking device while the furnace power is on.

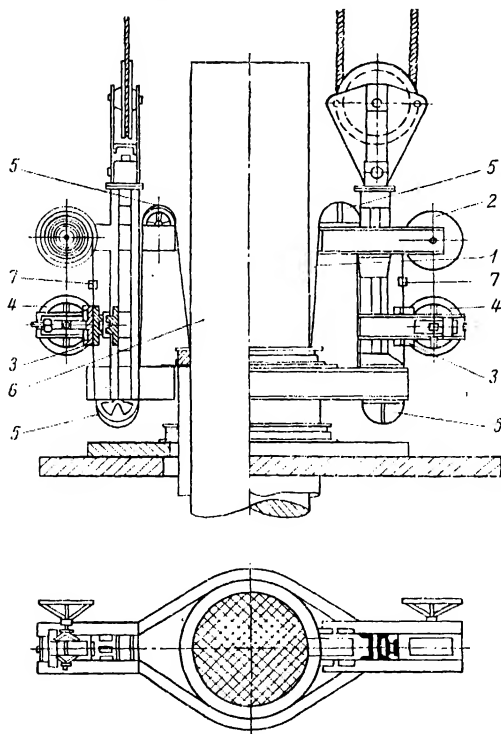


Fig. 23. Braking device for the slipping of electrodes

A self-baking electrode of an ore-smelting furnace is suspended from two steel bands 1 (Fig. 23) welded to the electrode shell 6. Each band is reeved through two iron blocks 5 to provide friction and through friction grip 3, actuated by wheel 4. Band 2 is reeled on an iron rod resting on two brackets. The band is chosen of a size capable of withstanding the weight of the electrode. For a 950-mm diameter electrode the band cross-section is 1.5×120 mm. As the

electrode is gradually fed down the band is welded to the shell. In working position the device is locked and the electrode travels together with the electrode holder. The electrode-slipping operation is performed as follows:

a) limiting blocks 7 are moved to the distance of electrode slipping, i.e., approximately 100 to 250 mm above the locking grip;

b) both grips are loosened simultaneously by rotating the wheels, and the electrode sinks by gravity when the electrode holder is equipped with spring-locking mechanism; when the electrodes are locked by setting screws the latter are loosened with socket wrenches and are retightened when the operation is over;

c) as soon as the electrodes have been slipped friction grips are tightened with the aid of the above-mentioned wheels.

A new spring-pneumatic device intended to mechanise the above operation was recently designed in the Soviet Union.

Its main elements are two grip rings (upper and lower) which lock and slip the electrode by means of compressed air, fed to and evacuated from rubber cylinders.

The lower grip is secured to the electrode shell while the upper one is stationary. The movement of the upper grip is ensured by hydraulic jacks or springs.

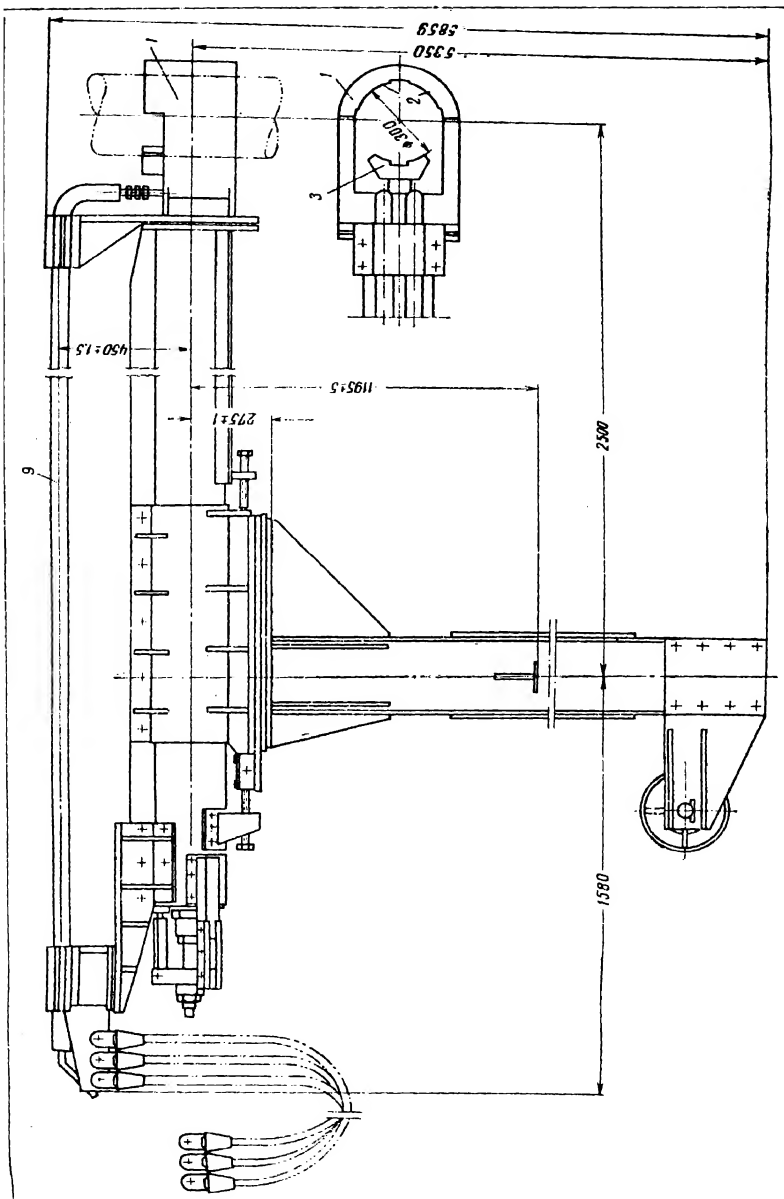
When an electrode is to be slipped air is let out of the cylinders of the lower grip and jack pressure is removed, after which the electrode sinks by gravity. The lower grip is then locked and the upper one is freed; the latter is lifted by means of jacks or springs and is again locked.

The chief advantage of this design is the possibility of remote-controlling the electrode-slipping procedure and the absence of bands welded to the electrode shells, thus ensuring a smooth surface for the electrode and a more reliable contact between the clamp and the electrode.

Furnaces with Electrode Holders Rigidly Connected to a Mobile Carriage

Electrode holders of this type are used in refining furnaces; they may be divided into several types: tong, chain or ring. Fig. 24 shows a tong-type electrode grip used for an electrode 300 mm in diameter. The electrode grip is provided with two contact clamps 1 and 2 with conducting buses 5 and water-cooling system 6, articulated joint 3, and coupling screw 4. The grip is tightened or loosened by a coupling screw.

An electrode grip of the chain type is shown in Fig. 25. This grip is provided with four clamps interconnected by articulated joints



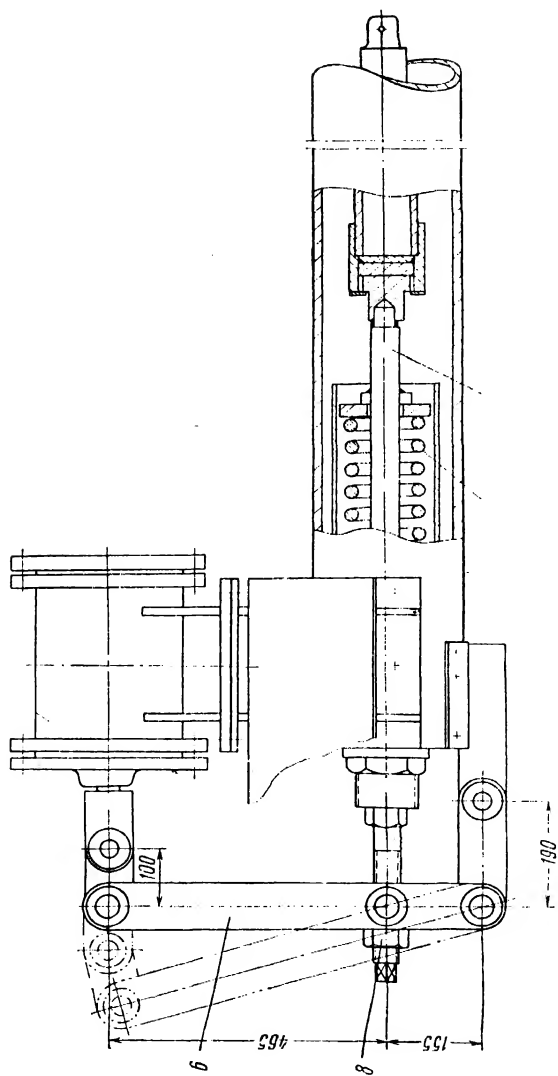


Fig. 26. Spring pneumatic clamp for a 300-mm electrode:
a—general view; b—control

and braced together by a screw. The provision of four clamps increases their surface of contact with the electrode.

The spring-pneumatic grip shown in Fig. 26, recently designed, is successfully operated; there are indications that this type of grip may displace all other designs of clamping devices.

The electrode is locked by clamps 1 (secured to ring 2) and clamps 3 fastened to stem 4. Clamping is brought about by stem 4 and springs 5 while declamping is effected by lever 6 with pneumatic

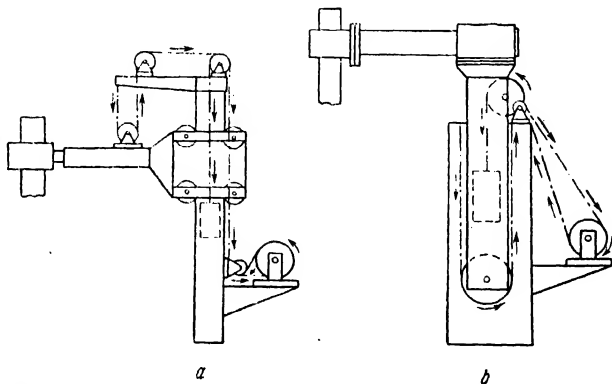


Fig. 28. Diagram of electrode-positioning mechanisms (arm type):

a—with mobile carriage; b—with telescopic support

drive 7. When compressed air is fed into the pneumatic drive cylinder the piston actuates stem 8; lever 6, moving leftwise, pulls stem 4, thus compressing spring 5 and freeing the electrode. The clamps are suspended from the ring secured to an angle bracket by means of bolts. Since the mechanism operates in high-temperature conditions, the electrode grip clamps are water-cooled. Current is fed through pipes 9 which simultaneously serve as water ducts.

When the furnace is operated with graphitised electrodes it is of prime importance to shorten the length of the working end of the electrode to avoid breakage. The spring-pneumatic grip for these electrodes was designed at the Chelyabinsk Ferroalloy Smelter (Fig. 27).

There are two types of travel mechanisms of bracket-type electrodes (Fig. 28): a stationary upright with a mobile carriage and a mobile upright connected to a carriage (telescopic upright).

In mechanism with a mobile carriage (Fig. 28, a) the horizontal arm supporting the electrode grip is connected to the carriage mov-

ing on wheels along the stationary upright. The stationary uprights more often than not are columns of a rectangular cross-section with counterweights located inside. The uprights are braced into a single framework.

In mechanism with telescopic uprights (Fig. 28, *b*) the horizontal arm is rigidly connected to the upright which moves along with the arm. In this case the mobile upright moves on guiding rollers installed in the stationary uprights; the latter are secured to the furnace shell.

In both cases the horizontal arm is either a thick steel tube reinforced by stiffening ribs or a welded box-like water-cooled beam.

A furnace equipped with telescopic electrode holders is smaller than that equipped with mobile carriage electrode holders. Moreover, electrode holders with telescopic uprights ensure better water-cooling of the arm, permit rapid adjustment of electrode circle diameter and allow the use of simple designs for the mechanised slipping of electrodes. The electric hoisting mechanism, however, is more massive and requires a more powerful electric motor.

Electrode carriages or mobile uprights are actuated by electric or hydraulic drives. The motors of electrode-hoisting mechanisms are installed behind the uprights or on the upper furnace platform. They are equipped with counterweights to reduce power consumption. The drives should be of a closed type (dust-proof body); electrode travel speed is 0.7 m/min. Carriage travel is adjusted by extreme position catches. The arrangement of mobile elements of an electrode-hoisting mechanism is shown in Fig. 28.

Cooling System

Temperature in the working zone of an electrode holder of a large ferrosilicon furnace reaches 400°C and may rise to 1000°C when an overfire occurs. Temperature in the working zone of an electrode holder is about 600°C during the melt-down period in furnaces working on the batch principle. Therefore, the electrode holder and conducting circuitry should be water-cooled to ensure normal operation.

Furnaces are provided with the following water-cooling systems:

- 1) cooling of current-conducting pipes and contact clamps;
- 2) cooling of electrode-holder ring;
- 3) cooling of electrode shell;
- 4) cooling of the surface of the supporting framework exposed to heat from the furnace-top.

Water fed into each water-cooling system from a post is subsequently returned to the header. Each feeding branch is equipped

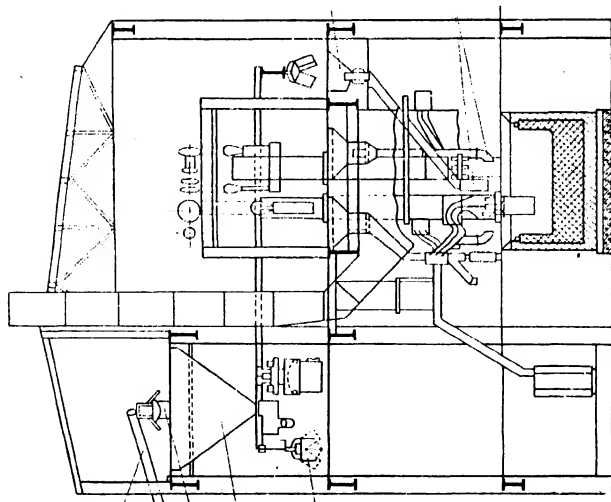


Fig. 29. Diagram of the charging system of a ferroalloy furnace:
 1—sloping conveyor; 2—horizontal conveyor; 3—furnace car; 4—weight car; 5—furnace hopper; 6—charging pipes

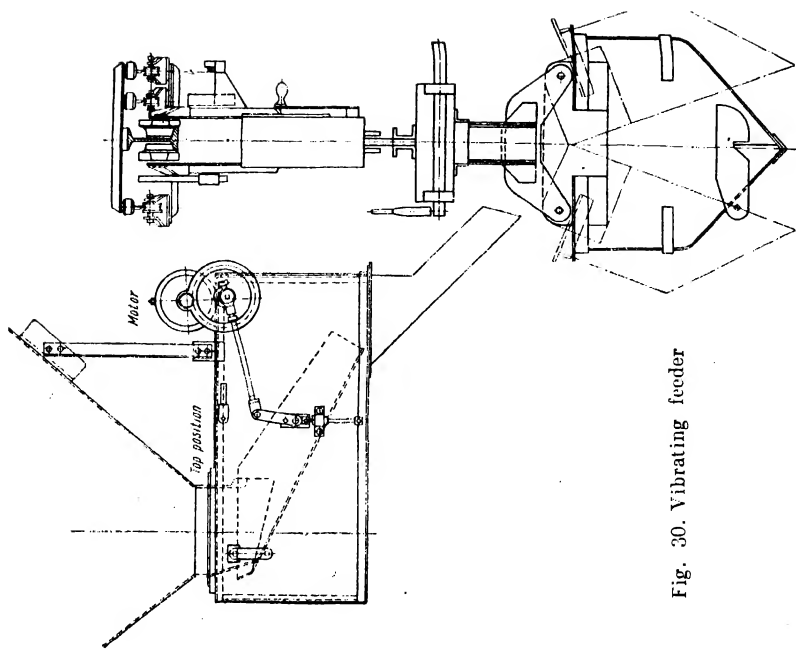


Fig. 30. Vibrating feeder

with a valve to regulate water flow. In the zone of flexible cables water is fed and collected with the aid of rubber hoses with thermal insulation.

Each electrode holder should be cooled independently. Clamp-cooling systems should also be independent and electrically insulated from all other cooling systems. Fig. 17 shows the clamp-cooling system diagram.

Contact clamps are cooled in pairs or in a series system, i. e., water out of current-conducting tube goes through a copper pipe into one clamp ("little lyre"), then through a second pipe to the second clamp ("big lyre"), then again through the "little lyre" into the current-conducting tube and after that into the header. Other furnace elements are cooled in the same way to ensure efficient use of the thermal capacity of water, i.e., to lower its consumption.

Over-all water consumption averages 5 m³/hr per 1,000 kVA of transformer rating. The temperature of the outflow should not exceed 50°C to prevent salt deposition on the inner surface of cooled elements.

To remove salt deposits it is recommended to flush the cooling system with a weak solution of hydrochloric acid. Water pressure in the feeding system should average 3 atm. It should be noted that cooling water carries away heat. L. I. Morozensky and G. M. Weinstein have calculated that, in the case of a 7,800-kVA furnace, the heat carried away by water amounts to 980 kW, with the half being lost through the cooling of contact clamps. Consequently, there has been a tendency of late to use heat-resisting elements instead of water-cooled ones wherever possible, or to utilise the heat of outgoing water. Furnaces requiring no cooling of electrode shells and electrode-holder suspension ring have been used successfully in recent years.

Mixture-Conveying and Charging Systems

Consumption of raw materials for the manufacture of certain grades of alloys exceeds 150 tons per day per furnace. Mechanisation of mixture conveying and charging is therefore of prime importance. Preparation and conveying of mixture in Soviet smelters is fully mechanised, but machine-charging has not as yet been solved for all alloys. Fig. 29 shows a raw-material conveying flow-sheet in use at most Soviet ferroalloy smelters.

Mixture prepared in the stockyard is brought by inclined belt conveyors to horizontal conveyors and thence by a tripper to furnace bins. The bin shutter is about 3 metres above the platform used for electrode jointing. The shutter is equipped with a vibrating tray feeder shown in Fig. 30.

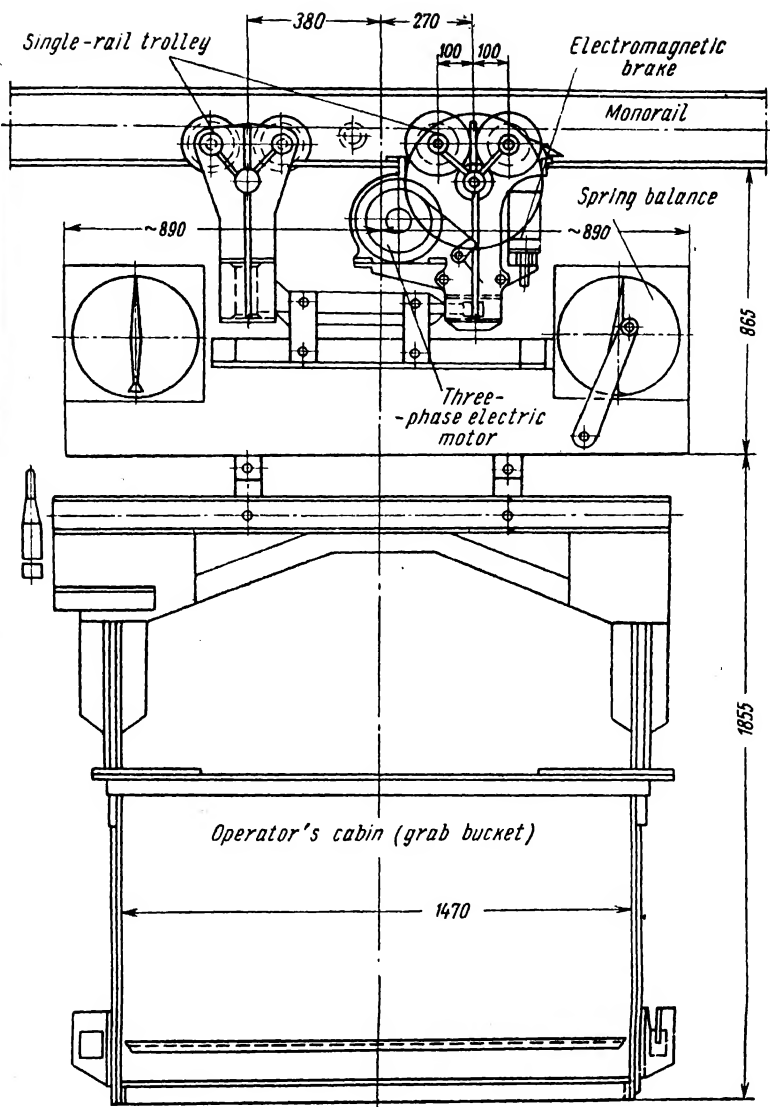
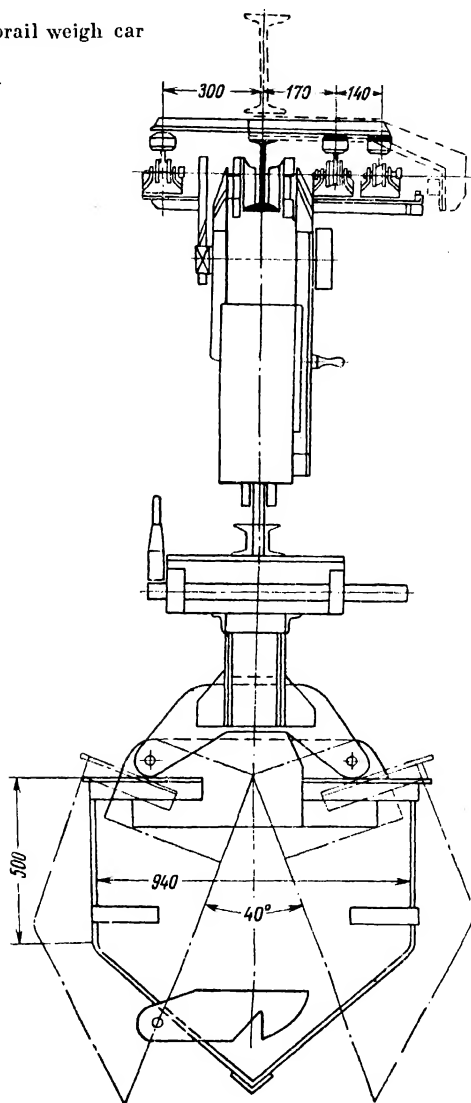


Fig. 31. Monorail weigh car



From the furnace bins the mixture is transferred to a monorail proportioning trolley equipped with spring scales. The carriage drive, shown in Fig. 31, is a telfer. The carriage moves along a monorail, stops at each bin and takes in the necessary amount of required mixture.

It then travels to the furnace pockets and empties the mixture into one of them through a drop bottom.

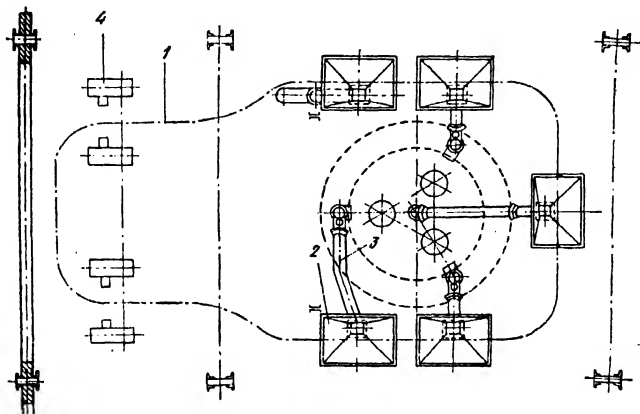


Fig. 32. Weigh car trackage:

1—car tracks; 2—furnace hopper; 3—charging pipes; 4—furnace bin feeder

The carriage is operated by push-buttons; a special device forces the carriage bucket to open over one of the pockets. Carriage trackage and the furnace pockets are shown in Fig. 32.

In the smelting of ferrochrome or ferromanganese the charge is fed directly to the furnace through pipes.

In the case of silicon alloys, the mixture in most cases is charged by means of a Pluiko charging machine schematically shown in Fig. 33.

The machine is mounted on a self-propelling car moving on rails around the furnace. The direction of shovelling may be adjusted to any point of furnace space by means of a rotating mechanism and by changing the inclination of the shovel. The materials are loaded from pockets into the machine bin and thence a pusher apportions them into the loading shovel. The throwing mechanism hits the loading shovel with the result that the mixture is thrown into the furnace. A pneumatic damper is provided to soften the impact of the loading shovel against thrust blocks.

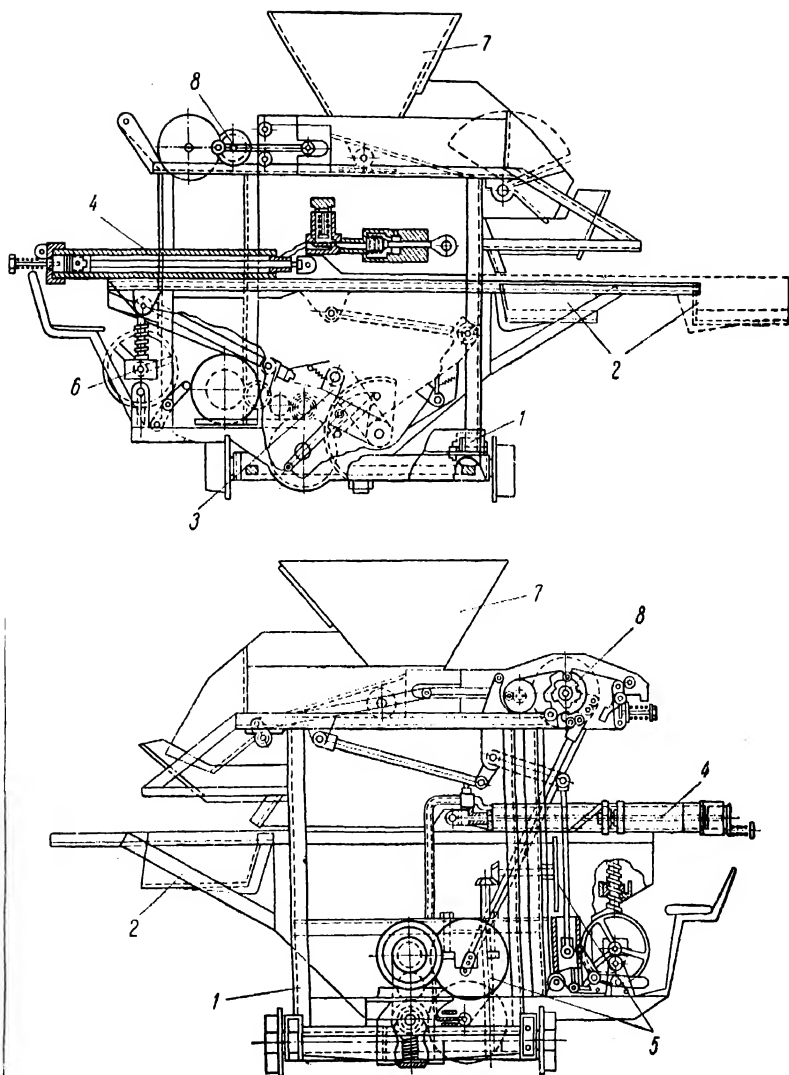


Fig. 33. Diagram of the Pluiko charging machine:

1—self-propelling car; 2—charging shovel; 3—pusher; 4—pneumatic damper; 5—vertical rotating mechanism; 6—pan-tilting mechanism; 7—mixture bin; 8—pusher mechanism for feeding mixture into the charging pan

The use of charging machines has facilitated the work of furnace-men, raised per-man output and made it possible to cut the number of operators. Charging by a Pluiko machine is illustrated in Fig. 34.

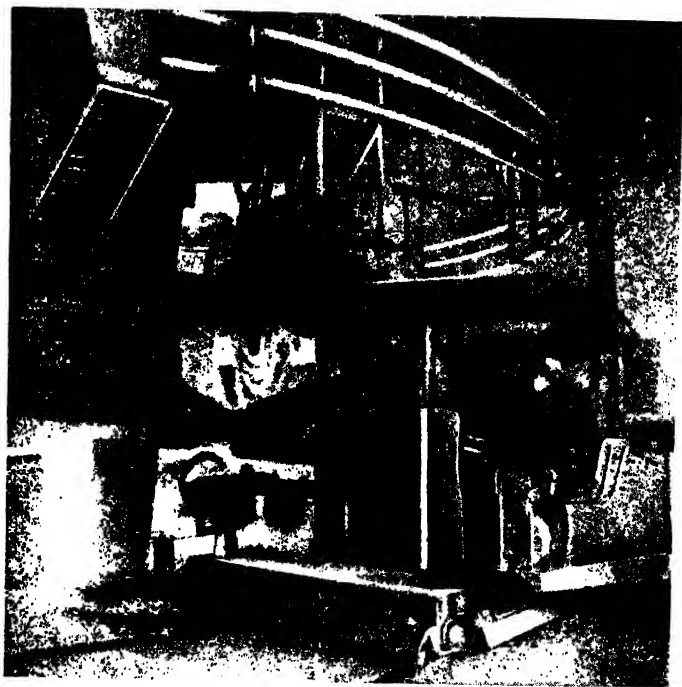


Fig. 34. Charging by means of the Pluiko machine

One of the possible systems of charging mixture materials into a rotating bath and closed-top furnaces in smelting ferrosilicon is shown in Fig. 8.

Exhaust and Conveying Ventilation

The ferroalloy smelting process causes considerable evolution of gases and dust. A hood with an exhaust duct rising 5 to 10 m above the smelter shop roof is installed over the furnace-top to remove gases (a forced exhaust system is sometimes used). Exhaust gases should be directed through dust-collecting systems. When

the escaping gases contain dust laden with oxides of valuable elements they are usually handled by electrostatic precipitators (Fig. 35).

Furnaces are provided with shields of diverse configuration to protect workers from heat radiation from the furnace-top or

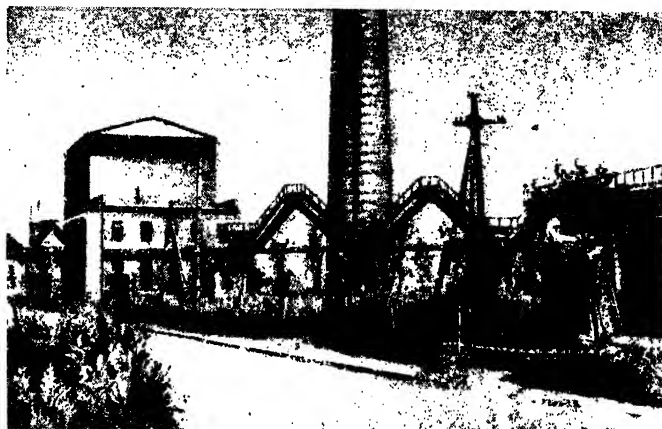


Fig. 35. Electrostatic dust precipitator of a ferromolybdenum smelter shop

melt surface; the shields are secured between the hood and the furnace shell. Moreover, clean moist air is fed to the working platform by means of powerful fans.

IIA-11-type open disk fans are available around the furnace where the operators work. The gases which appear when alloys are tapped are caught by the hood and carried off by an exhaust fan.

Electrical Tapper

Alloys are tapped regularly several times a shift. It is not always possible to open a taphole with a tapping iron and in such cases it is opened ("burned through") by means of an electric arc generated by a special device. This device may also be used to remove accretions of metal and slag in the taphole. A schematic diagram of such a device is shown in Fig. 36.

The device is connected to the terminals of a phase winding of a transformer; voltage across these terminals is equal to that between the winding and the furnace hearth. The device comprises interrupter switch *I*, by means of which current from bus *13* is fed

to holder 4 and electrode 5 through buses 12 and 2 and flexible bands 3. The holder is equipped with insulated pad 7, handle 6 and counter-weight 10; the holder is secured to rotating block 9 by means of stem 8. The shunt is switched off or on by wheel 11. The taphole is burned open by an iron bar or directly by the electrode.

Equipment Maintenance and Overhauling

Furnace equipment is inspected thoroughly each shift by the chief furnaceman, his helper and the shift fitter.

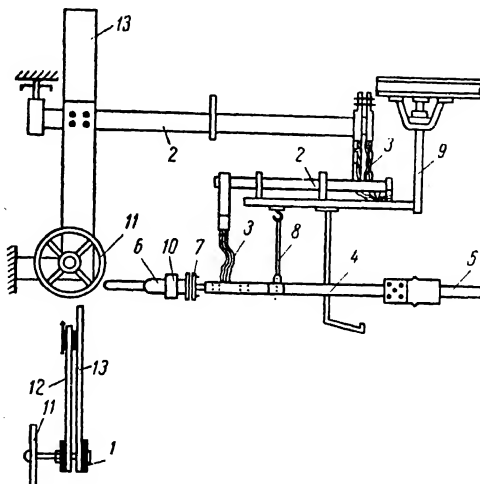


Fig. 36. Tapping apparatus

One of the main factors for the reliable service of flexible cables and the electrode holder is the normal functioning of the water-cooling system. In the event of water failure the furnace should immediately be switched off to eliminate the possibility of breakdowns.

Care should be taken to prevent foreign matter from penetrating into the water header when the furnace is in operation (this causes clogging).

The chief furnaceman and shift personnel should immediately eliminate hot-spits, for they frequently cause the breakdown of electrode-holder elements.

When operating electrode-suspension and travel mechanisms, thorough check should be made of pulley insulation, the condi-

tion of the glands which shield the electrode deck from gas and dust, the verticality of electrode suspension and the condition of the electrode-hoisting winch and its steel cables.

The temperature of the buses in the furnace zone should not exceed 70°C. Bus sections exposed to furnace-top heat radiation should be shielded. Low-voltage bus-bars and cables should also be protected from short circuits caused by random contacts and furnace spittings. When one of the flexible cables or some other element of the low-voltage circuit heats up it is necessary immediately to eliminate the causes.

Maintenance periodicity depends on the operating conditions of the equipment. Overhauls are usually carried out according to the following time-table:

a) large ore-smelting furnaces for silicon alloys, carbon and foundry ferrochrome and ferromanganese are shut down for maintenance repairs every four or five months for one to three days and once every five to ten years for a general overhaul for ten to twenty days;

b) refining furnaces are shut down for maintenance repairs once a month for 24 hours and once or twice a year for six to eight days for a general overhaul; furnaces smelting ferrotungsten are overhauled once every five years.

General overhauls include mechanical equipment repair, transformer inspection, replacement of low-voltage circuitry and, as a rule, furnace relining.

During preventive maintenance shut-downs special attention should be paid to inspection and necessary repairs of electrical cables with obligatory check-up of all electrical contacts, water-cooling systems, electrode-holder elements, electrode-suspension mechanism and hoisting winches. Charging and proportioning equipment is overhauled, ventilation is checked and, if necessary, repaired. Inspection of oil breaker, oil pump, furnace transformer, secondary switching and functioning of measuring devices is compulsory. Lining is partly replaced when necessary.

Chapter 3

FERROALLOY FURNACE ELECTRICAL EQUIPMENT

Conversion of Electric Power into Heat in Ferroalloy Furnaces

In an arc furnace, electric power is converted into thermal. Electric current flowing through the arc, melt and charge, generates the following amount of heat in a given time interval:

$$Q = I^2 R t \text{ joules,}$$

in which

Q — quantity of heat, joules;

I — furnace current, amperes;

R — total resistance of arc and melt, ohms;

t — time, hours.

1 joule = 0.239 calories.

This formula may be rewritten as follows:

$$Q = 860 \frac{I \times E}{1,000} t \text{ kcal,}$$

where Q — quantity of heat, kcal;

I — furnace current, amperes;

E — furnace voltage, volts;

t — time, hours.

Furnace Power Supply Circuit

Ferroalloy electric furnaces as a rule operate on three-phase alternating current.

A typical electric supply circuit is shown in Fig. 37.

Shown in the diagram are: 1—high-voltage interrupters through which power is supplied to the furnace by a substation double-bus distribution system; they are blocked in such a way that power may be fed only through one bus system at a time; 2—metering current transformers energising ammeters and wattmeters' current coils; 3 and 5—current transformers energising protection and automation relays; 4—oil breaker; 6—voltage measuring transformer connected to the high-voltage circuit through high-voltage protecting device 7 with resistors; 8—furnace transformer; 9—low-voltage circuit; 10—electrodes and 11—furnace hearth.

Furnace Electrical Operating Conditions

A ferroalloy furnace is a powerful electrical unit. Each technological process and each ferroalloy furnace require definite operating conditions, i.e., a definite relationship between furnace electrical characteristics (rating, current, voltage and corresponding electrode circle) which provide maximum furnace productivity with minimum power consumption per ton of alloy. Determination of these operating conditions, called optimum operating conditions, is a very important practical problem. Furnace electrical operating duty has an important effect on performance results of ferroalloy smelting.

There are two different groups of technological processes in electric furnace ferroalloy-smelting: continuous and batch-smelting (refining).

In continuous processes power is applied uninterruptedly and the furnace has a constant power input. The burden is charged by small batches, while the metal is tapped periodically. Smelting of silicon and its alloys, production of carbon-bearing ferrochrome and ferromanganese are continuous processes.

Processes requiring batch-smelting, such as the production of refined ferrochrome, are distinguished by variable power input at different periods and by unstable operating conditions.

Current distribution in a ferroalloy furnace varies with the character of the process.

In continuous processes, a gas-filled chamber called the "crucible" forms under each electrode (Fig 38). Its walls are red-hot charge materials, its bottom—the melt, its top—the electrode. The volume of this crucible increases with power input. The arc burns calmly in the closed space and relatively low voltage is required to strike it and to keep it burning.

Electric current forms arcs between the electrode and the crucible walls as well as between the electrode and the melt. A certain amount of current is shunted between the electrodes through the mixture. Power developed in the crucibles by the current is the main component of active furnace power. The heating of the mixture outside the reactive zone is not economical considering the negli-

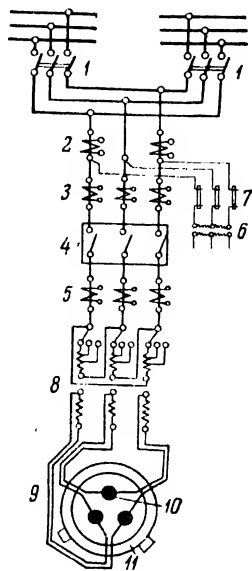


Fig. 37. Primary commutation circuit of a ferroalloy furnace

gible height of the furnace and the big radiation surface of the mixture. Practically, there is always a trend to operate with a "cold furnace-top", i.e., to secure heat generation in the reactive zone and to lower useless and even harmful (because it complicates operating conditions and increases the losses of the reducing agent) heat generation at the furnace-top.

In furnaces operating on the batch-smelting principle (Fig. 39),

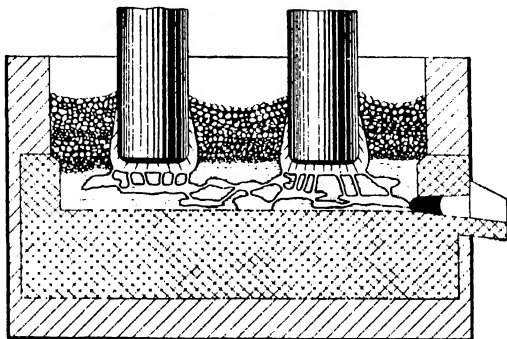


Fig. 38. Diagram of laboratory of a furnace smelting ferrosilicon

all power is practically developed by the arcs and by current flowing from one electrode to another through slag and metal. Power generated in both slag and metal is useful because metal and slag have to be kept hot to ensure normal furnace operation.

Working Voltage

No definite method of choosing operating voltage for ferroalloy furnaces has yet been elaborated. Operating data show that the magnitude of working voltage depends on furnace rating and the type of alloy smelted. Electric furnaces with transformer, rating of 2,500-3,500 kVA and secondary voltage of 150-350 V are generally used for the refining processes. The smelting of ferroalloys by continuous processes requires more powerful electric furnaces, with transformer rating of 7,000 kVA and over. In continuous processes the arc is closed in the crucible and a relatively low voltage is needed to keep it up. Modern furnaces with transformers rated at 10,000-12,000 kVA are provided with voltage at the tip of an electrode equal to 60-65 V. Maximum electrode immersion is necessary to ensure the maximum utilisation of the heat generated in the arc. This may

be attained by lowering secondary voltage. However, with the power input being constant, the secondary voltage reduction causes a sharp increase in electric losses in the low-voltage circuit. The use of higher secondary voltage decreases losses in the low-voltage circuit, but entails higher heat losses through the furnace-top, since

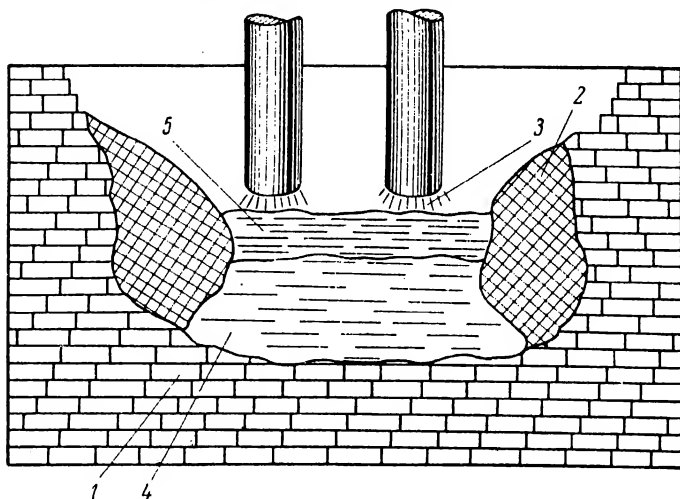


Fig. 39. Diagram of the hearth of an electric arc furnace for smelting refined ferrochrome:

1—magnesite brick; 2—protective layer; 3 - electric arc zone; 4 - alloy; 5 - slag

the length of the arc is increased while electrode immersion in the charge is reduced.

For continuous ferroalloy smelting processes an optimum electrode immersion of about 1-1.6 m has been justified by practice; this ensures satisfactory performance results. In furnaces of 7,500-12,500 kVA this has been achieved by keeping secondary voltage at 130-180 V.

Furnace Power Factor and Efficiency

Furnace power factor or cosine phi ($\cos \varphi$) is the ratio of true power to total apparent power, i.e.,

$$\frac{P_{\text{true}}}{P_{\text{apparent}}} = \cos \varphi, \text{ wherefrom } P_{\text{true}} = P_{\text{apparent}} \times \cos \varphi.$$

$\cos \varphi$ increases with the rise of voltage, this being explained by the growth of active resistance as the result of an increase in arc

resistances which are practically a true ohmic load. The true power of the furnace, which is equal to true power minus active electrical losses in the circuit, increases with the rise of voltage, i.e.,

$$P_{\text{effective}} = P_{\text{true}} - P_{\text{true losses}}$$

The ratio of useful furnace power to total true power is called furnace electrical efficiency η_e , i.e.,

$$\eta_e = \frac{P_{\text{effective}}}{P_{\text{true}}}$$

Furnace efficiency or electrical efficiency η_{power} is the product of two values

$$\eta_{\text{power}} = \eta_e \times \eta_{\text{thermal}},$$

where η_{thermal} is furnace thermal efficiency characterised by the performance of the furnace as a thermal apparatus. The greater the rating of the furnace the higher its η_{thermal} , for the relative value of thermal losses drops when heat is used more efficiently.

Specific Power Consumption

Specific power consumption, i.e., power consumption per ton of salable alloy is an over-all index of furnace design, operation methods and personnel's skill.

Specific power consumption per 24 hours is equal to

$$a = \frac{W}{Q},$$

where a —specific power consumption, kWh/ton;

W —power consumption per 24 hours, kWh;

Q —alloy output per 24 hours, tons.

Furnace Transformer

Furnace Power Rating

Larger transformers are more economical and are used in all cases where technological operating conditions permit it. The furnaces now employed for ore-smelting processes are provided with 40,000 kVA transformers. Transformers of 2,500-3,500 kVA are used for refining processes, provided furnace operation is not hampered. The required transformer power rating may be calculated according to the following formula:

$$P_{\text{kVA}} = \frac{Q \times a}{24 \cos \varphi \times k_1 \times k_2 \times k_3},$$

where P_{kVA} —transformer rating, kVA;

Q —furnace productivity per 24 hours when smelting a given alloy, tons per day;

a —specific power consumption, kWh/ton;

$\cos \varphi$ —furnace power factor;

k_1 —transformer load factor;

k_2 —furnace down-time factor;

k_3 —factor which takes into account all circumstances which complicate furnace operation (for example, voltage drop in the main).

Operating data show that for a powerful furnace the following values may be attributed to the above-mentioned factors: $k_1=0.98$; $k_2=0.96$; $k_3=0.95$ and $\cos \varphi=0.9$.

Principle of Operation and Design of Furnace Transformers

A transformer is an electromagnetic device which serves to convert alternating current of a given voltage to alternating current of a different voltage.

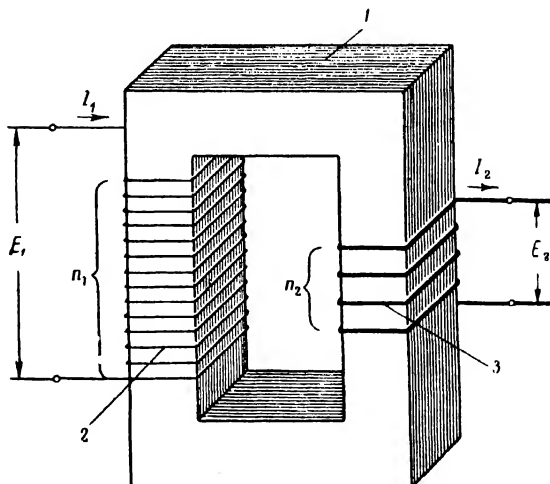


Fig. 40. Schematic diagram of a transformer

The transformer consists of steel core 1 (Fig. 40) which is an assembly of insulated sheets of electric steel (with 4% Si and low hysteresis losses) 0.35-0.50 mm thick, and two copper wire windings:

primary winding 2, to which energy is fed, and secondary winding 3, to which the furnace is connected.

Transformers work on the following principle: an alternating current flowing through the coils of a primary winding produces an alternating magnetic flux which, in turn, induces an electromotive force in the secondary winding. When the number of turns of the

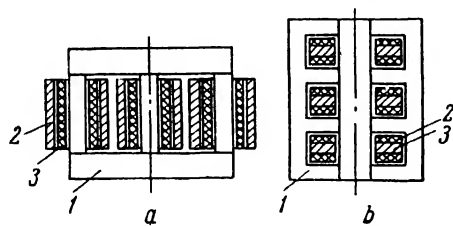


Fig. 41. Typical diagrams of furnace transformers:

a—core-type; b—shell-type;
1—magnetic circuit; 2—primary winding; 3—secondary winding

secondary winding is lower than that of the primary winding the transformer is called a step-down transformer as the voltage in the secondary winding is lower than the primary (supply) voltage. When the turns ratio is inversed the transformer is called a step-up transformer.

Let E_1 , I_1 and n_1 represent voltage, current and number of turns of the primary winding of a transformer and E_2 , I_2 and n_2 —the same for the secondary winding.

Transformation factor K of a transformer is the ratio of the number of turns of the primary winding to that of the secondary winding or the ratio of primary e.m.f. to the secondary e.m.f. when the transformer is not under load:

$$K = \frac{n_1}{n_2} = \frac{E_1}{E_2} (1).$$

Energy losses in a transformer are low and its efficiency equals 0.98 and over, and it may be assumed that energy input of a transformer and energy supplied to the furnace are equal:

$$E_1 \times I_1 = E_2 \times I_2.$$

Then, taking into account the expression (1):

$$K = \frac{n_1}{n_2} = \frac{E_1}{E_2} = \frac{I_2}{I_1}.$$

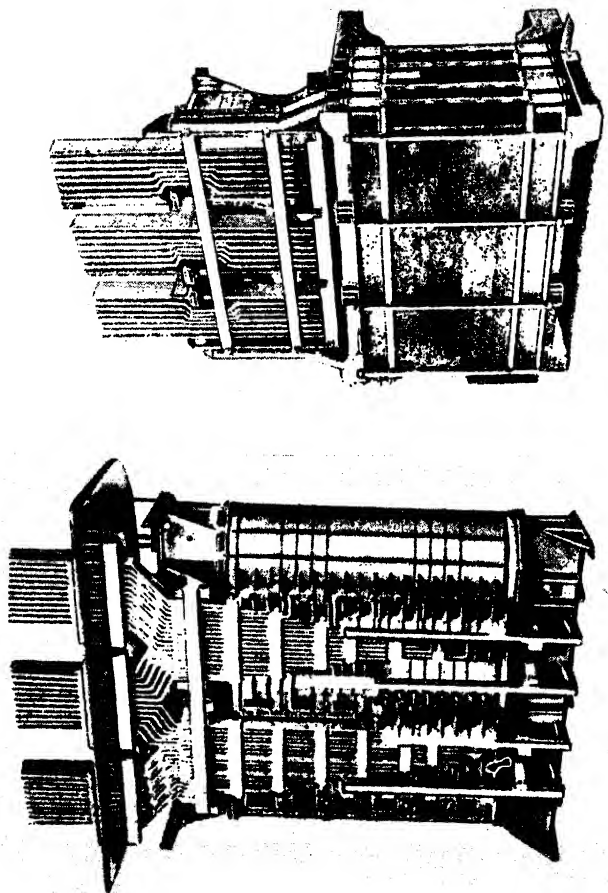


Fig. 42. Transformers taken out of shells:
a—core-type; b—shell-type

Thus, the secondary current of a transformer is equal to:

$$I_2 = K \times I_1.$$

A three-phase transformer or a bank of three single-phase transformers are used for the transformation of three-phase currents. A three-phase transformer consists of three primary and three secondary windings arranged on a common iron core—the magnetic circuit.

In the case of large furnaces (over 7,500 kVA) it is better to have a group of three single-phase transformers, this permits to reduce the length of the low-voltage circuit and improves the power factor and electrical efficiency. In this case it is necessary to provide only one reserve single-phase transformer for a group of furnaces; this

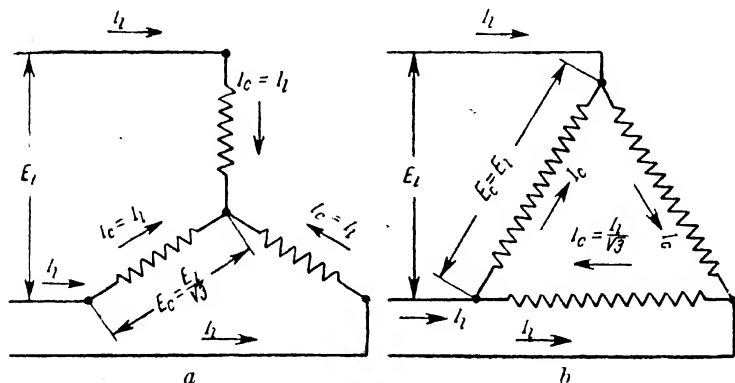


Fig. 43. Connection of windings:
a—star-connection; b—delta-connection

makes replacements or overhauls considerably cheaper and more convenient than with three-phase transformers.

According to magnetic circuit design and winding arrangement, transformers are divided into two types: the shell-type and the core-type transformers (Fig. 41). In shell-type transformers the winding is surrounded by a magnetic circuit and its servicing entails greater difficulties than that of the core-type transformer windings. Core-type transformers require less steel than those of the shell type.

Shell-type transformers better withstand the dynamic effect of currents, and the cooling of their windings is superior (especially in the case of the single-phase types) to that of the core type.

Fig. 42 shows both types of transformers (without the shell). Soviet plants manufacture mainly core-type transformers.

The windings of a three-phase transformer or of the single-phase transformers making up a three-phase bank may be either star-connected (Y-connection) or delta-connected (Δ -connection). A variety of relations will exist between the e.m.f.s and the currents in the windings, depending on the type of connection used (Fig. 43).

Star-connection is a method of connecting the three transformer windings in which one end of each winding is connected to a common neutral point. Mesh- or delta-connection is the method when the end of the first coil is connected to the beginning of the second, the end of the second coil is connected to the beginning of the third

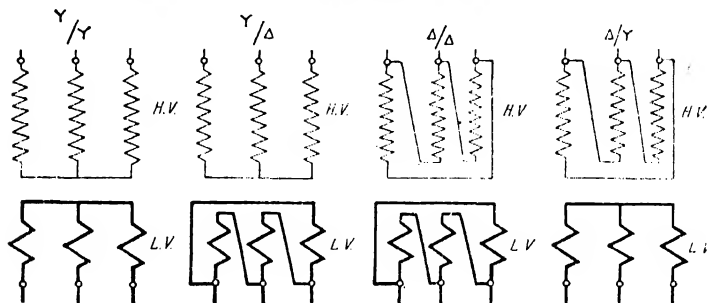


Fig. 44. Diagram of winding connections of three-phase transformers

and the end of the third coil is connected to the beginning of the first. Line wires are connected to the terminals of phase windings. There is no neutral point in a delta-connection.

Let E_l , I_l represent voltage and current in the line (line voltage and line current), E_c , I_c voltage and current in any one transformer winding (coil voltage and coil current).

In power engineering it is a known fact that in Y-connection the coil voltage is $\sqrt{3}$ times less than the line voltage ($\sqrt{3} \approx 1.73$) $E_c = \frac{E_l}{1.73}$. Coil current in this case is the same as in the line, i.e., $I_c = I_l$.

In Δ -connection we have $E_c = E_l$ and current flowing in the line passes through two transformer windings and we have $I_c = \frac{I_l}{1.73}$.

Y- or Δ -connection may be used for primary as well as for secondary transformer windings and, consequently, there may be four types of connection for three-phase or a bank of three single-phase transformers (Fig. 44).

A furnace transformer is distinguished by the taps in the secondary windings. The number of taps depends on their function and

4) high transformation ratio as the furnace installation is supplied with high-voltage current while the furnace itself is fed with low-voltage current.

Characteristics of furnace transformers for standard series of ferroalloy furnaces are given in Table 2.

Table 2

Ferroalloy Furnace Transformer Characteristics

Type	Rated power, kVA	Input voltage, V	Voltage taps, V	Electrode current, A
ЭТМН 5000/10	2,500	10,500	178-149-127- -111-89	8,100-9,580- -11,550-11,750- -13,000
ЭТНН 5000/10	3,500	10,000	303-189-270- -240-216	6,550-7,000- -7,500-8,430- -8,960
ЭТНН 10000/10	4,500	10,500	178-149-128- -112-89	14,600-17,400- -20,300-23,200
ЭТНН 10000/35	9,000	10,000	158-148.5-140.5- -133-126.5-120- -110-105.5	32,800-35,000
ЭТНА 10000/35	10,500	10,000	158-148.5-140.5- -133-126.5-120- -110-105.5	38,400
ЭТН 15000/10	12,500	10,000	178.5-167.5-158- -149-141.5	40,400-43,150- -45,000
ЭТН 15000/10	12,800	10,000	185-172.5-161- -144-130.5	40,000
ЭОНН 8200/10	5,500 one-phase	10,000	210-203-196- -189.5-183.5- -178-172.5- -167.5-162.5- -158-154-150- -146-142-139- -135.5-132.5	26,200-33,850

Low-Voltage Circuit

The circuit from the terminals of the secondary side of furnace transformer is characterised by considerable currents and the large size of conductors; this low-voltage circuit, because of its negligible length, is sometimes called "short main".

The low-voltage circuit of a ferroalloy furnace usually consists of three principal sections (Fig. 47): bus assembly 1, flexible section (string) 2 and current conducting pipes 3 which are connected to electrode holder contact clamps.

An alternating current of great magnitude, of the order of tens of thousands of amperes, flows through the low-voltage circuit, causing

strong magnetic fields which induce an alternating e.m.f. in the circuit. The induced e.m.f. always moves in the direction opposite to that of the current which induces it and thus creates additional electric resistance to the flow of current through a conductor—inductive or reactive resistance. About 10% of the power input is

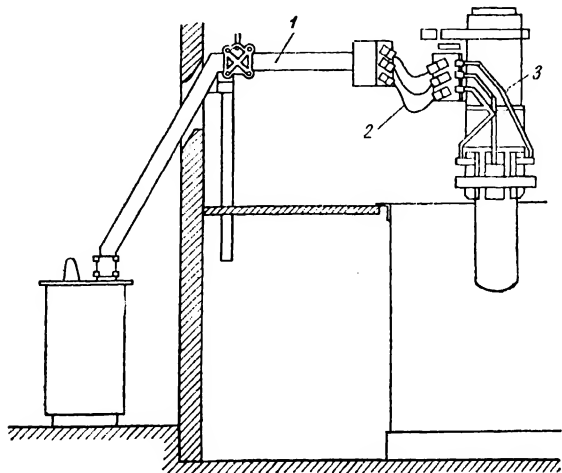


Fig. 47. Diagram of a low-voltage circuit

lost in the low-voltage circuit of a ferroalloy furnace and a minimum of inductive resistance should be aimed at in low-voltage circuit design. This may be attained by:

- 1) maximum shortening of the circuit;
- 2) packaging of conductors (closest possible arrangement of conductors);
- 3) choosing optimum cross-section of the conductor, taking note that self-induction grows less with the increase in perimeter to cross-section ratio;
- 4) placing the conductors as far as possible from massive steel structures.

The magnetic field and the induced e.m.f. are decreased to a great extent when the three cables of a three-phase main are arranged close to each other. As it is difficult closely to arrange large three-phase conductors of high current-carrying capacity, each phase cable is made up of a certain number of smaller conductors. All three-phase conductors should then be interleaved in such a way as

to place conductors of different phases as close to each other as possible. An over-all diagram and method of arrangement of such a three-phase circuit are shown in Fig. 48. This interleaving cannot be carried out with flexible buses and current-carrying pipes as it will result in increased inductive resistance.

The more closely the buses are arranged and the greater their height the less is the reactivity of the system. Copper water-cooled

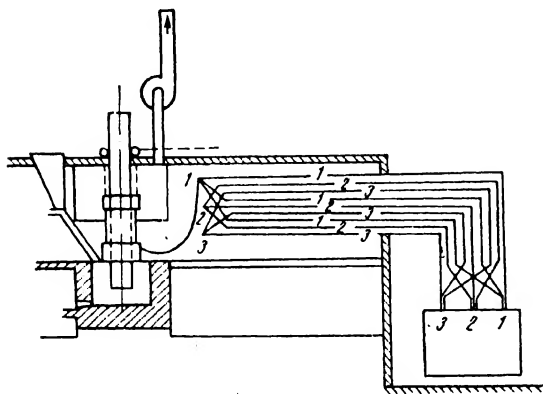


Fig. 48. Interleaving of buses in a ferroalloy furnace



Fig. 49. Compensated three-phase power supply circuit made of oval pipes

pipes are often used instead of bus-bars when large currents are involved; oval-shaped pipes are used to provide minimum spacing intervals (Fig. 49).

The secondary terminals of furnace transformers are made up of a large number of single wires to facilitate interleaving in the system (Fig. 42).

A very effective method of decreasing the inductive resistance of low-voltage circuits is the bifilar arrangement of conductors, in which currents flow in opposite directions in two adjacent wires; thus, the magnetic field created by one current is neutralised by the opposite current.

Star- or delta-connection of transformer windings should be effected outside the transformer in order to obtain a bifilar low-voltage circuit.

An arrangement of conductors from the transformer to the furnace

is shown in Fig. 50. In this arrangement the circuit is delta-connected near the furnace. Bifilarity is complete from *A* to *B* while the section from *B* to *C* may give considerable asymmetry. Wiring diagrams shown in Figs. 51 and 52, in which mesh-connection is

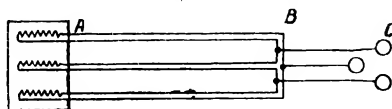


Fig. 50. Compensated power supply circuit with a delta-connection realised at the furnace

effected directly on the electrodes, permit to obtain systems with well-balanced magnetic fluxes.

Power transfer from one phase to another may occur as the re-

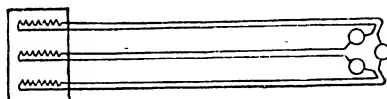


Fig. 51. Compensated power supply circuit with a delta-connection realised at the electrodes

sult of a difference in inductive resistance in various asymmetrically spaced windings; these are the so-called "wild" and "dead" phase effects. In this case the electrode connected to the "wild"

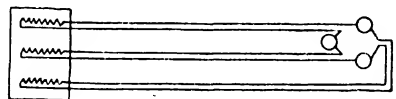


Fig. 52. Compensated power supply circuit with a delta-connection realised at the electrodes

phase winding overheats, its operation is rendered difficult and heat losses increase, while the charge melts down very slowly beneath the "wild" phase electrode.

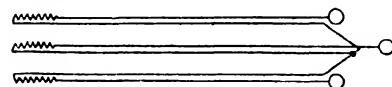


Fig. 53. Compensated power supply circuit with a star-connection between the electrodes

An example of a balanced star-connected circuit is shown in Fig. 53. When three single-phase transformers are used such a circuit permits individual voltage control for each of the three furnace electrodes. A perfectly symmetrical current system in the use of three single-phase transformers is obtained with the arrangement shown in Fig. 54. With such a wiring no circuit asymmetry and, consequently, no "wild" or "dead" phase effects are observed.

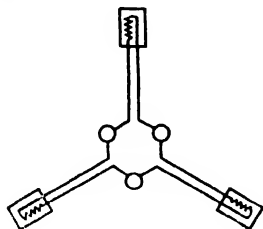


Fig. 54. Compensated power supply circuit with transformers symmetrically arranged around the furnace and a delta-connection realised at the electrodes

In this case, the furnace single-phase transformers are mounted on the working platform level. This sharply decreases the length of conductors and energy losses and raises electrical efficiency.

Furnaces used for refining processes are of a lower rating and operate with a higher voltage and corresponding lower currents. Therefore, the above-mentioned considerations concerning the reactivity of low-voltage circuits are of no great significance for these furnaces.

Commutation and Protective Apparatus, Metering Instruments and Furnace Control

Commutation equipment consists of a number of apparatus intended for switching the furnace on or off. These devices are installed in the primary-voltage system of a transformer, since switching would be difficult on the secondary side with current values of tens of thousands of amperes. Oil breakers are generally in use for switching electric furnaces on or off because the electric arcs caused by air-switching would unavoidably destroy (fuse) the contacts.

An oil breaker (Fig. 55) is a tank filled with transformer oil. Porcelain bushings 1 are secured to cover 5. Current is supplied by copper rod 2 (which passes through the bushing) to the left pair of contacts 10, then through a transverse copper blade 9 to the right pair of contacts 16, and goes out through a rod in the second bushing.

Each phase is provided with two upper stationary contacts and two lower ones, which are mobile. The lower contacts of the all three-phase windings are secured by contact holders 3 to crosspiece 4.

When the latter is pulled down each of the three current circuits (on all three phases) is disrupted at two points.

Rapid disconnection of the oil breaker is ensured by strong springs 12. Spark extinguisher contact 11, when operated, is pressed against the mobile contact. The cover is secured to the uprights or

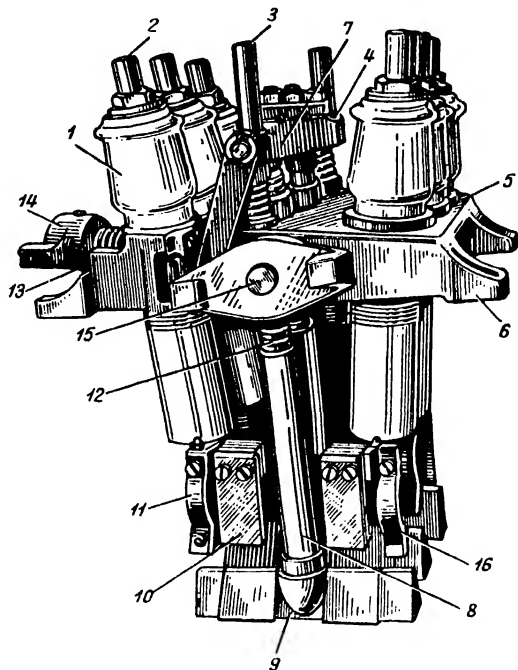


Fig. 55. Oil breaker

brackets by means of lugs 6. Contact holder rods are insulated by bakelite tube 8. The oil breaker is normally switched on or off by means of wheel 15 or by remote control with the aid of a solenoid coil. The tank is secured by bolts to lugs 13 of the cover; the tank may be lowered for the replacement of contacts or oil by special winch 14. Buffer spring 7 is provided to soften the crosspiece impact against the cover.

A cavity communicating with ambient atmosphere through a safety valve is left unfilled to prevent the explosion of gases which evolve when the arc is extinguished in the oil breaker. Oil serves

to extinguish the arc as well as for the insulation of conductors. It gets dirty on account of frequent disconnections, its insulation becomes unreliable and it should be changed.

When the oil breaker is in the off position, its bushings and stationary contacts, connected to the energy source, remain under high voltage. Breakers installed between the oil switch and the supply

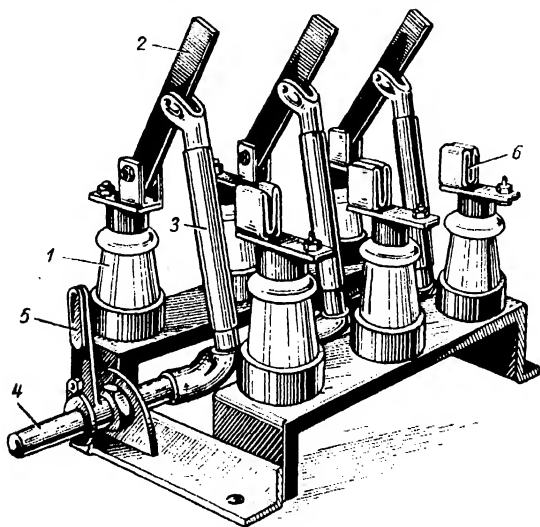


Fig. 56. Air-break three-phase disconnector

main serve to disconnect high voltage and to provide safe access to the oil breaker.

A breaker (Fig. 56) consists of supporting insulators 1, blade contacts 2, drawbar 3, axle 4, handle 5 and stationary contacts 6. This breaker is switched on or off only when the oil breaker is in the off position.

In electric furnace installations the furnace transformer is protected against all overloads. The following protective systems are usually provided:

- 1) gas protection by means of a gas relay;
- 2) overload relay;
- 3) short-circuit relay;
- 4) thermal relay;
- 5) grounding against the insulation failure of the high-voltage winding.

The gas relay (Fig. 57) is mounted on the sleeve which connects the transformer shell to the conservator, and is normally filled with oil to the full. The relay is provided with two mobile floats, one atop another. As long as the relay is filled with oil the floats are afloat and the built-in mercury contacts are open.

Any failure within the transformer (insulation puncture, short-circuit in the winding) generates heat and, consequently, causes the decomposition of oil and insulation. Evolving gases

force out the oil in the relay; the oil level falls, causing the contacts of the first float to close and to actuate a signal; then, if precautionary measures are not taken, the second float sinks and its contacts close the circuit for the automatic disconnection of the oil breaker.

Overload and short-circuit protection is provided with the help of electrical relays which disconnect the circuits when the current exceeds a predetermined value. Signal relays are of the electromagnetic DT-520 type; overload protection is provided by induction time-lag relays of the HT-81 type; instantaneous disconnection—over-current cut off—is provided with electromagnetic relays of the DT-520 type.

In its simplest form, such a relay is a coil (solenoid) with an internal mobile core. When the current exceeds a preset value the core undergoes retraction and closes the signal relay contacts, then the contacts of another relay for disconnecting the oil breaker. Dependent time-lag relays are generally used in overload protection circuits; these relays disrupt the circuit following a certain time-lag after the current overload. The greater the current overload, the less the time-lag. Relays mentioned above reliably protect the transformer against overloads and reduce to the minimum the number of disconnections.

Thermal relays of various designs operate when the oil temperature in the transformer rises above the permissible level.

A complete set of furnace-control instruments usually comprises:

- 1) three ammeters and three voltmeters (each type of meter per winding) for measuring winding currents and voltages;
- 2) ammeter and voltmeter for measuring current and the voltage of the transformer primary circuit;
- 3) three-phase recording MWmeter;

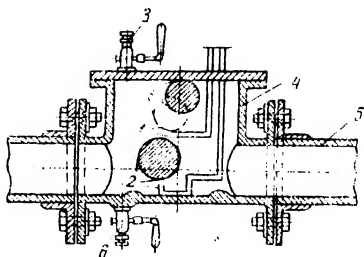


Fig. 57. Gas relay:

- 1—signal contacts; 2—disrupting contacts; 3—tap for gas sampling; 4—relay frame; 5—conservator connection pipe; 6—tap for testing relay

4) three-phase wattmeter.

The MWmeter and the wattmeter are usually connected to the high-voltage side of the transformer and measure energy losses in the transformer.

All meters are energised by means of instrument transformers which provide for:

1) safety in servicing meters;

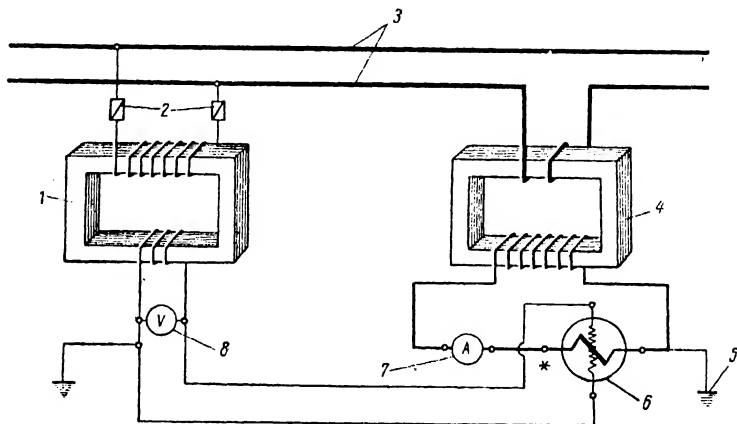


Fig. 58. Connection of electrometering instruments to the high-voltage circuit:

1—voltage transformer; 2—high-voltage fuses; 3—high-voltage main; 4—current transformer; 5—protective grounding; 6—wattmeter (counter); 7—ammeter; 8—voltmeter

2) possibility of a distant positioning of metering instruments from the point of measurement, which makes it possible to concentrate all the meters in one panel;

3) possibility for standardising the types and sizes of metering instruments.

A wiring diagram for electrical measuring instruments is shown in Fig. 58.

A thermocouple for hearth temperature and a thermometer for transformer oil temperature are also available.

The furnace is operated from the control panel on the basis of instrument readings. It is possible to control furnace operation both automatically and manually. The current may be increased by lowering the electrodes and decreased by raising them. Manual control of electrode travel is left as a reserve to be used in case automatic control fails. Moreover, it is indispensable for certain technolog-

ical operations: power input raise when smelting grade Xp.0000 ferrochrome, baking of electrodes, etc.

A wiring diagram of the XЭМЗ (Kharkov Electromechanical Works) automatic control system for providing the stability of power input is shown in Fig. 59. Regulator action is based on the principle of balance between the current flowing through the electrode and the arc voltage. The differential or balance relay is

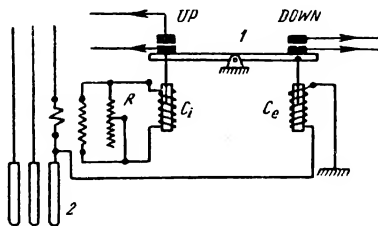


Fig. 59. Diagram of automatic regulator

provided with two coils: C_e (voltage) and C_i (current). The iron cores of the coils are connected to current-carrying rocking arm 1, which closes the contacts UP and DOWN when the coil is retracted by the corresponding coil. Current coil C_i is inserted into the circuit of the secondary winding of the instrument transformer (current metering) of the corresponding furnace winding and its current is proportional to the current of the above-mentioned winding. Voltage coil C_e is connected to the buses which supply current to electrode 2, i.e., in parallel with the arc. Current proportional to the arc voltage flows through coil C_e . When the values of arc current and voltage are correspondingly equal to preset values, their thrust on the cores is equal, and the rocking arm is in horizontal position (equilibrium) and the contacts UP and DOWN are open. When the thrusting actions of the coils become unequal as the result of a change in operating conditions, the coil that is stronger pushes its core upwards; this closes the corresponding contact with the aid of the rocking arm. Thus, when the current value rises, the pushing action of coil C_i overcomes the counteraction of voltage coil C_e and the rocking arm closes the contact UP. The hoisting of an electrode leads to the decrease of current and to a simultaneous rise of arc voltage. As soon as the thrusts of the coils become equal, the contacts open and the electrode movement ceases until the preset operating conditions are disturbed once again.

Current automation has proved its worth for silicon alloy furnaces. For slag processes it is better to use automatic regulators with amplidyne.

Chapter 4

ELECTRODES

Role of Electrodes and the Requirements They Should Fulfil

Electrodes serve to supply electric current to the working space of a ferroalloy furnace and constitute one of the main elements of its design. The physical and chemical properties of the electrodes essentially affect the quality of the smelted products and the performance data of ferroalloy production.

Continuous self-baking electrodes (Söderberg electrodes), which have supplanted carbon electrodes wherever carburisation of alloys or their dilution by iron is permissible, are widely used at present. Carbon electrodes are not employed in the ferroalloy industry in the Soviet Union while graphitised electrodes are used in the production of decarburised ferrochrome, metallic manganese and crystal silicon.

Electrodes should possess high electrical conductivity, sufficient mechanical strength, high oxidation temperature and low disintegrating capacity and low cost.

Generally used are ring electrodes. The shape of the electrode should be geometrically perfect to ensure a reliable contact between the electrode and the clamp.

Soviet ferroalloy smelters use electrodes ranging from 250 to 1,400 mm in diameter.

The physicochemical properties of graphitised electrodes are as follows:

Ash content, %	0.1-1.0
Apparent specific gravity, g/cm ³	1.50-1.70
True specific gravity, g/cm ³	2.20-2.24
Porosity, %	28-30
Mechanical strength (compression), kg/cm ²	160-300
Specific electrical resistance, ohm·mm ² /m	8-12

The characteristics of anode paste for the self-baking electrodes used in various smelters (carbon electrode characteristics are given for comparison) are shown in Table 3.

Table 3

Characteristics of Anode Paste

Smelters	True spec. gravity, g/cm ³	Apparent spec. gravity, g/cm ³	Porosity, %	Mechanical strength, kg/cm ²	Electric conductivity, ohm·m
Chelyabinsk	1.91	1.38	30	107-298	67-80
Kuznetsk	1.88	1.34	29	218	101.5
Zaporozhye	1.80	1.50	20.7	280-320	60-70
Aktyubinsk	1.92	1.28-1.43	27-35	87-258	72-123
Carbon electrodes	1.9-2.05	1.48-1.65	20-22	200-450	35-55

The table above shows that the quality of self-baking electrodes is almost equal to that of carbon electrodes.

Permissible current density equals¹, A/m²:

For carbon electrodes 5-11

For graphitised electrodes 13-28

For self-baking electrodes 5-7

¹ Maximum current values correspond to electrodes with smaller diameters.

Manufacture of Electrodes

The electrode manufacture flowsheet shown in Fig. 60 applies to all sorts of electrode products; it may vary with the raw materials used as well as with the technological equipment available.

Electrode manufacture is characterised by numerous operations of considerable duration which, in the manufacture of graphitised electrodes, take about two months.

Carbonaceous materials used for the manufacture of electrodes (anthracite, coke, graphite and coal-pitch) are stored in special, usually closed store rooms to prevent them from becoming damp and soiled.

Anthracite is the basic component of carbon electrodes and anode paste for self-baking electrodes. It has a high carbon content, great mechanical strength, and low ash content, and is of low volatility. Anthracite which has gone through thermal treatment in special furnaces is called thermoanthracite. The use of thermoanthracite in electrode-manufacturing works makes it possible to substitute drying for calcination.

The cost of anthracite and thermoanthracite is correspondingly 11.90 and 27.00 rubles a ton.

Anthracite is considered suitable for the manufacture of thermo-anthracite (electrode and foundry) when it is strong and thermally stable, possesses high specific gravity, dense homogeneous structure, fine-grained fractures, and is free of lamellar pieces and noticeable cleavage:

Ash content, %, maximum	5
Sulphur content, %, maximum	2.5
Strength (shatter index), over	70
Thermal stability index, over	60
Specific gravity, over	1.5
Size of lumps, mm	50-120
Fines (lumps less than 50 mm), %, below	5

Moisture in the Donets coalfield anthracite deliveries is set by the standards 1'OCT (the State Standard of the U.S.S.R.) 180-48, 1'OCT 222-49, 1'OCT 243-50.

Anthracite should contain no lumps of pyrite or barren rock.

The quality specifications for electrode thermoanthracite deliveries prescribed by 1'OCT 4794-49 are as follows:

Moisture, %, mean	0.2
Ash content, %, maximum	5.0
Sulphur content, %, maximum	1.9
Volatile content, %, maximum	0.3
Lumps larger than 40 mm (trommel test), %, minimum	70
Specific electric resistance, ohm·mm ² /m, below	1,000
Size of lumps, mm	25-100
Fines content (lumps less than 25 mm), %, below	5

Coke is one of the chief raw materials used for the manufacture of electrodes, particularly graphitised. The most valuable are petroleum and coal-tar pitch cokes.

Petroleum cokes are solid products obtained from petroleum residues by a high-temperature process. Pitch coke is obtained by carbonising coal pitch. Pitch and petroleum coke is distinguished by a low ash content (below 1.0%) and low volatile matter content.

Coal coke is the product of destructive distillation of some grades of coals in coke ovens; it has a higher ash content (from 8 to 18%) and is used for the manufacture of carbon electrodes (foundry coke); the so-called coke fines are used as a charge component in graphitising furnaces.

Coke characteristics are given in Table 4.

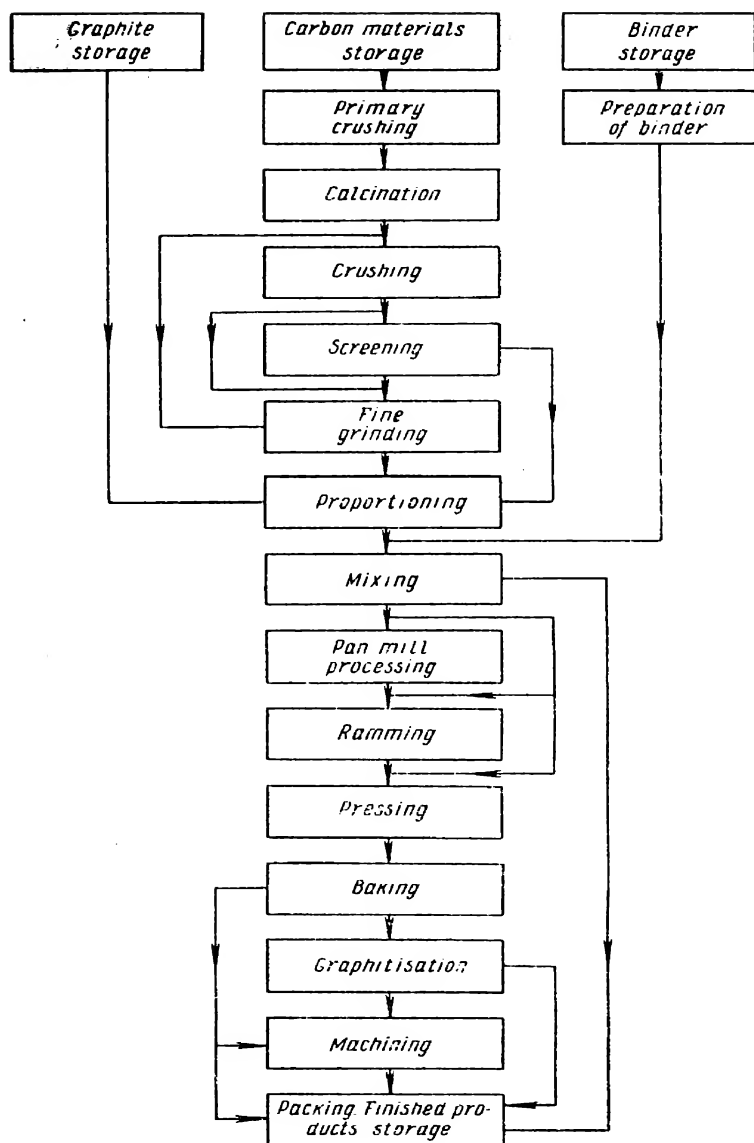


Fig. 60. Flowsheet electrode manufacture

Table 4

Technical Specifications of Various Grades of Coke

Data	Petroleum coke	Pitch coke	Foundry coke	Ductile coke	East Karakans coke	Fine coke nut
						Mean
Moisture maximum	0.0	0	4.0	—	4.0	11.0
Ash maximum	0.5	1.0	1.4	12.0	15.0	12.16
Sulphur content maximum	1.0	0.4	1.4	—	0.7	0.7
Volatile matter % maximum	7.0	1.0	1.0	—	1.7	2.4
Iron maximum	0.08	0.12	—	—	—	—
Phosphorus maximum	—	—	—	0.014	—	0.04
Lumps (>25 mm) minimum	1.0	4.0	Below 10 mm	1.0	5.0	Below 10 mm
			4.0			10.16

The price of coke, rubles per ton

Time metallurgical coke	11.70	Petroleum coke	30.10
Foundry coke	17.50	Pitch coke	71.70-7.00

Graphite when added to the anode paste enhances its plasticity, makes pressing easier (thus permitting to obtain denser products), and increases electrical conduction, thermal conductivity and heat resistance. Ash in graphite should not exceed 10%. The price of graphite is 150-200 rubles per ton.

Table 5

Pitch Characteristics

Data	Soft pitch	Acidic pitch	Alkalic pitch
Temperature of softening °C	10-10	6-7	1-70
Total carbon content maximum	12.0	25.0	20.0-2.0
Ash maximum	0.3	0.9	0.3
Moisture % maximum	0.4	0	0.3
Volatile matter %	Not standardised		60-65

Coal pitch is a solid, lustrous black in colour. Its specific gravity ranges from 1.245 to 1.299. Its technical specifications are given in Table 5.

Coal pitch is used as a binder; it costs 29 rubles a ton. Pitch is sometimes stocked in liquid form in specially heated hoppers called pitch conditioners. Another type of binder—coal tar—is stored in underground heated reservoirs in the binder storeyard. This viscous liquid is a by-product of coke.

Electrodes

Raw materials are initially crushed by roll- and jaw-crushers (Figs. 61 and 62) and then calcinated.

Calcination is thermal treatment of carbonaceous materials at high temperatures and without air access. All carbonaceous materials used for the manufacture of electrodes (with the exception of graphite) are subjected to this treatment. This operation increases apparent and true specific gravities, electrical conductivity and mechanical strength, and reduces the volatile content. It is carried out in furnaces of various types; a retort calcinator widely used in the U.S.S.R. is shown in Fig. 63.

Calcinated materials are crushed in crushers of various types and in ball mills, and are subsequently classified with the aid of rotary sieves and rocking or vibrating screens.

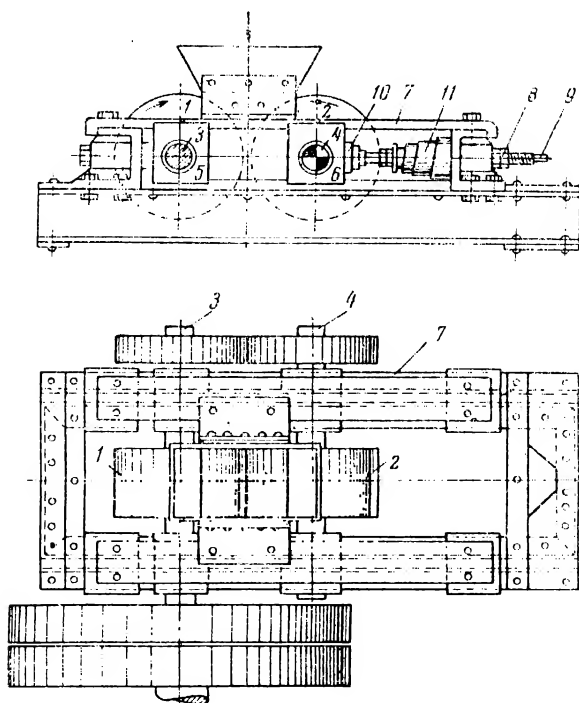


Fig. 61. Roll crusher:

- 1, 2—rolls; 3, 4—shafts; 5—stationary bearings; 6—mobile bearings;
7—frame; 8—nuts; 9—bracing bolts; 10—washers; 11—spring

Conditioned materials are precisely proportioned and conveyed to the mixing machines. A homogeneous mixture called electrode or anode paste is obtained in these machines; it goes either for the subsequent manufacture of electrodes or is considered to be a finished product.

The paste meant for the manufacture of electrodes is conveyed to a large mixing machine which acts as a conditioner and in which the temperature of the paste is levelled out.

Anode paste products are obtained with the aid of hydraulic presses by two methods: 1) pressing with closed impression dies, and

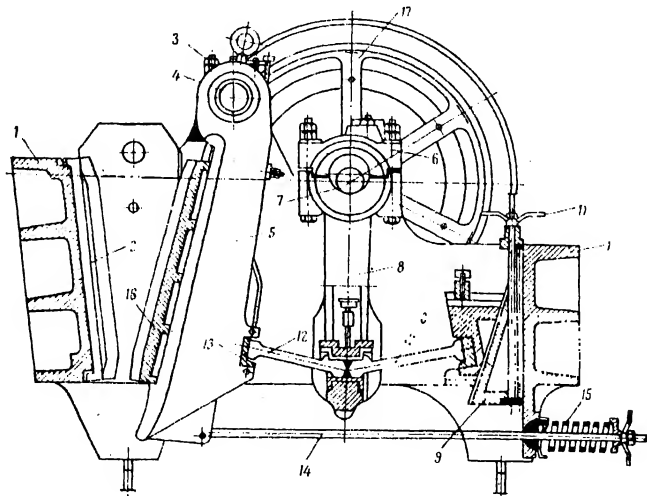


Fig. 62. Jaw crusher:

1—frame; 2—fixed jaw; 3—bearings; 4—axle; 5—mobile jaw; 6—bearings; 7—shaft; 8—pitman; 9, 10—wedges; 11—bolt; 12—toggle; 13—toggle seats; 14—tension rod; 15—spring; 16—jaw plate; 17—flywheel

2) extrusion of the paste through a nozzle (the broaching method). Preliminary pressing is usually done directly in the press container which increases the compactness of electrodes and levels out the pressure in the operation. Pressed ("green") electrodes are water-cooled on special rolls.

"Green" electrodes acquire mechanical strength as the result of baking; their electric conductivity and thermal resistance are drastically increased. Baking is effected at 1300°C in neutral atmosphere which preserves the electrodes from being burned and deformed. Coke fines are used as a filling. Baking and subsequent cooling lasts

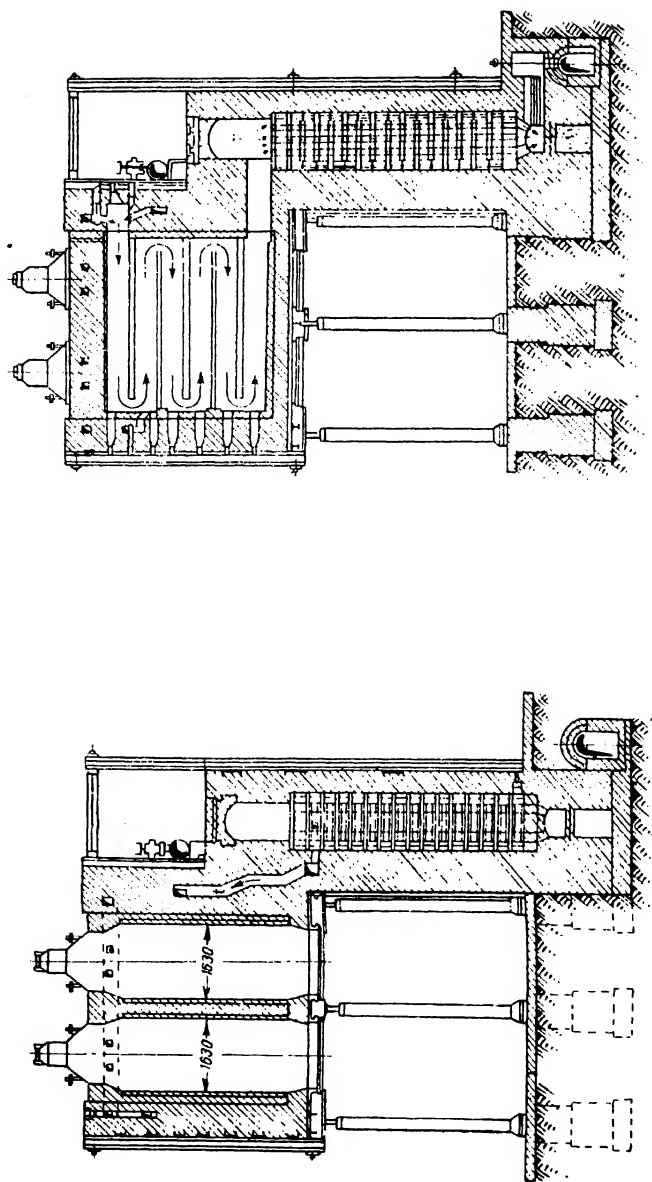


Fig. 63. Retort-calcinating oven:
a—section through the retorts; b—section through the channels

from 16 to 30 days; during this period the binding materials are coked and the electrodes acquire properties mentioned above.

The baked stock meant for the manufacture of graphitised electrodes is graphitised in electrical resistance furnaces at 2500°C ; the resistance elements are the electrodes themselves and the filling. The furnace (Fig. 64) consists of rectangular trough 1, lined with firebrick, and two end walls 2, through which pass current-carrying electrodes 3. A mixture of carbonaceous materials is rammed in the trough to form a tamped hearth. The electrodes subject to the graphitisation are laid on the hearth between electrodes 3, perpendicular to the furnace axis. One side wall 5 is made of firebrick while the other is of concrete blocks (sectional). The space between the

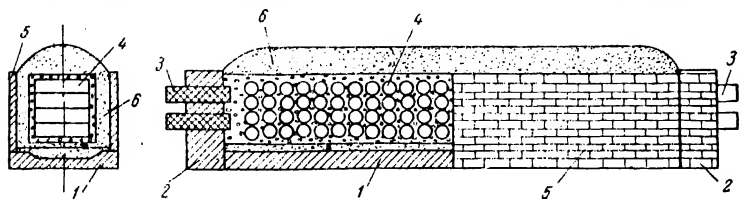


Fig. 64. Diagram of a graphitising furnace

electrode charge and the side walls is filled with heat-insulating material 6. A layer of insulating material also covers the electrodes.

As the result of graphitisation the products acquire valuable graphite properties: sharp increase in electrical conductivity, heat conductivity and chemical stability, diminution of hardness; the electrodes are readily machined, this ensuring the accuracy of nipple joints.

An additional operation—soaking—is effected to improve the physicochemical properties of graphitised nipples and to increase their compactness and toughness.

Baked stock is soaked with pitch in autoclaves at $200\text{--}300^{\circ}\text{C}$ and 5 kg/cm^2 for several hours, depending on the cross-section area of the semiproduct. After that the latter is graphitised.

Baked carbon and graphitised electrodes are then machined; this operation consists of turning the outer cylindrical surfaces of the electrodes and the electrode ends, and of threading nipple joints.

A nipple joint comprises a cylindrical nipple with a screw thread and a nipple recess in the butt-end of the electrode. Nipples are threaded from special stock of both high compactness and mechanical strength (Fig. 65).

Conic thread is often used for carbon electrodes; one end of the electrode is threaded with a conic nipple, while the other is a corresponding socket.

Electrodes are expensive because of this complex technological flowsheet and the high cost of raw materials, the cost coming to 123-170 rubles per ton for carbon and 244-335 rubles for graphitised electrodes. The higher cost of graphitised electrodes is explained by the necessity of effecting an additional operation—graphitisation—which consumes a large amount of electric power (approximately 6,000 kWh/ton).

Self-Baking Electrodes

The major advantages of self-baking electrodes in ferroalloy smelting are the following: the possibility of manufacturing large-diameter electrodes, their low cost, the possibility of adding joints without interrupting power input and the possibility of their on-the-spot manufacture.

A self-baking electrode is a metallic shell made of 1 to 2 mm sheet steel and filled with anode paste.

The manufacture of a shell section consists of the following operations:

- a) making of angle cuts in steel sheets;
- b) rolling of sheets along the cylinder generatrix, bending of sheet edges inside along the cylinder radius and bending of triangles;
- c) acetylene or projection welding of rolled sheets with an intermittent (not continuous) seam so that the gases evolve freely when the electrode is baked. Otherwise the gases may damage the electrode shell. The ribs may be made separately from the shell and then welded onto it.

The cylinder obtained (with the ribs bent inside) is approximately 1,400 mm long.

To facilitate jointing, special cuts are made in each section and the shells are somewhat tapered (the difference in diameters being 5 to 6 mm).

Inner ribs are intended to increase contact surface between the shell and the anode paste, to provide adequate adhesion and to make the electrical conductivity and mechanical strength of the electrode more reliable.

The number of ribs ranges from four to eight, depending on the electrode diameter; the rib height is increased correspondingly from 100 to 250 mm. There are not less than four triangles bent on each rib (the so-called "tongues"). These "tongues" strengthen adhesion between the shell and the anode paste.

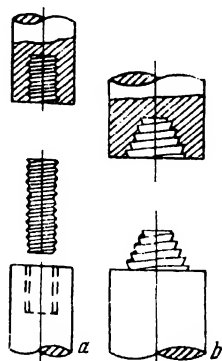


Fig. 65. Electrode nipples

The shell itself is a casing for the anode paste. It protects the latter against oxidation, provides passage for the electric current from the electrode arm to the baked section of the electrode, and intensifies heat transfer to the upper, unbaked section of the electrode.

Electrode paste is prepared in accordance with the above-mentioned technological flowsheet in electrode-manufacturing works or in special shops at the ferroalloy smelters.

The various manufacturing methods give the following ratio of electrode-paste components (in %):

Anthracite or thermoanthracite	37	38	55
Electrode-machining tails or metallurgical coke	38	38	21
Pitch	—	24	24
Tar pitch	25	—	—

The best electrode paste is obtained from medium pitch.

Electrode paste should conform to the following specifications:

- 1) ash content—maximum 10%;
- 2) volatile matter content—from 14 to 18%;
- 3) the paste should be well mixed, homogeneous in composition and free of any dirt or foreign inclusions.

In summer the volatile matter content should be limited to 13%.

Electrode paste is made in the form of briquettes weighing not more than 40 kg each.

Electrode paste should be transported and stored in conditions precluding its contamination.

The softening point of electrode paste is 40-70°C. It should retain the mechanical strength sufficient for its processing in a crusher, up to the above-mentioned temperature.

The electrode shell is filled with electrode paste without disconnecting the power supply. The paste is loaded as a solid at least once a day.

The size of anode-paste lumps emptied into the shell should not exceed 200 mm for large furnaces and 150 mm for smaller ones.

The electrode paste is filled to a level 4 to 5 m above the clamps.

The electrode top should be covered with a lid to avoid dust, since it may lead to subsequent electrode breakage.

Separate shell sections are jointed by means of gas-welding (continuous seam around the shell circumference) while the furnace is in operation.

The electrode shell should be so mounted that the ribs of the jointed sections were aligned with the electrode ribs; it is desirable to weld the ribs of the upper shell to the ribs of the lower shell section.

In welding shells, one must observe the following conditions:

- 1) see to it that the welded shell section is in a vertical position;
- 2) make sure that in its maximum elevation the electrode shell

does not interfere with the stationary framework and cannot be hit by an overhead crane.

In furnaces equipped with braking devices steel bands are welded to the electrode shell with a bilateral seam 100 mm long and at intervals of 200 mm in large furnaces and with a unilateral seam 50 mm long and at intervals of 100 mm in the smaller ones.

A special platform is mounted atop the furnace to facilitate electrode joint-making and filling.

The heat evolving from the furnace softens the paste and enables it to fill the electrode shell compactly.

Temperature for the baking of electrodes should be:

25°C when the paste is solid;

70°C when the paste is completely softened and melts into a solid block;

300°C when solidification sets in;

700°C when solidification ends with the evolution of the remaining volatile matter.

Temperature changes occurring in the electrode paste and the way in which temperatures are distributed in an electrode during the electrode-baking procedure in large ferrosilicon furnaces are schematically shown in Fig. 66.

The temperature rises to 50-70°C at 2-2.5 m from the lower tip of the clamps; by this time the paste is completely softened and melts into a compact block, thus increasing the heat conductivity of the paste. The temperature reaches 300°C at the level of the upper tip of the clamps and paste solidification sets in. The electrode is finally baked directly beneath the clamps where the temperature rises to 700-800°C.

Correct operation requires that the electrodes should not be baked above the clamps, for a satisfactory "clamp-electrode" contact is obtained only when the electrode enters the clamps in a softened condition and is "moulded" by them.

The cooling of electrodes should be watched as carefully as that of clamps, for an abnormal increase in the temperature causes premature baking of the paste.

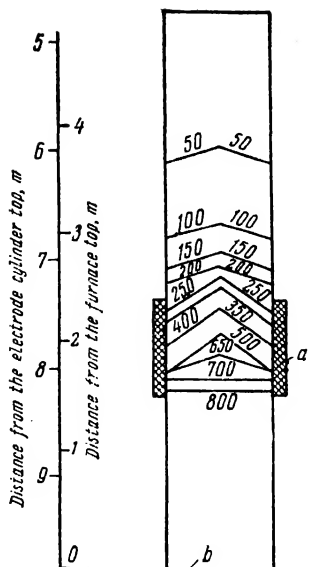


Fig. 66. Diagram of temperature zones in a self-baking electrode

When the electrode begins to bake above the clamps the operation of the open-disk fan, as well as the electrode cylinder and the gland gasket, should be checked.

Graphitised electrodes are jointed as follows:

- 1) the threaded socket of the upper electrode section is cleaned (preferably with compressed air);
- 2) the nipple is screwed in to the utmost;
- 3) bracing hoops are put on a new electrode and a lifting plug is carefully screwed into the latter's socket;
- 4) the lower nipple socket of a new electrode section is thoroughly cleaned;
- 5) the electrode is conveyed to the furnace by an overhead crane and mounted above the electrode to be jointed;
- 6) the new electrode is lowered to a point where its nipple enters the lower nipple socket of the new electrode section. The electrode should then be rotated around its axis; it will thus be unscrewed from the plug and screwed onto the nipple.

The screwing-off operation should be completed with two sharp twists of a wrench or lever;

7) the bracing hoops should be taken off as soon as the jointing is done.

Operation of Electrodes

The slipping of graphitised or self-baking electrodes in furnaces which are not equipped with brakes is effected in the following manner:

- 1) furnace power supply is disconnected;
- 2) the electrode is fed down until it rests on the furnace bottom or until it rests on a freshly charged mixture (when extra-low carbon alloys are smelted); it may also be supported by a crane;
- 3) clamps are pressed back and the electrode holder is lifted usually 300 to 400 mm;
- 4) the clamps are secured, electrodes lifted and furnace power reapplied.

Before being slipped the electrode is thoroughly air-cleaned to ensure reliable electrical contact between the clamp and the electrode.

There are various failures leading to breakages one may encounter in working with graphitised electrodes.

Poor centering of electrode holders may cause electrodes to break in the clamps when the latter are locked. Poor contact in the nipple joint is quickly discovered due to sparking and local incandescence; poor contact may also lead to electrode breakage. Electrode breakage may be caused by incorrect charging or negligence as a result of direct impact on the centre section of the furnace when it is being

cleaned or when attempting to lift an electrode frozen in the melt, etc.

In normal operating conditions the slipping of self-baking electrodes is effected with unbaked electrodes, this ensuring reliable clamp-electrode contact. When baking electrode above the clamps, it should be slipped so as to provide contact with the unbaked electrode.

Slipping of self-baking electrodes with the power on was described above.

It should be noted that the use of a spring lock in electrode holders permits to slip electrodes without disconnecting the power supply. Electrodes are slipped 100 mm each shift. This increases furnace efficiency. Moreover, spring clamps ensure uniform clamp pressure, thus improving operating conditions as a result of identical contact between the clamp and the electrode for all clamps.

In operating with spring clamps the welding of the breaking device band to the electrode shell should be carefully checked, as the disruption of bands leads to breakdowns, for the electrode in this case is held only by the bands and if the latter fail it will drop into the furnace.

After each electrode slipping the time and the height of lowering should be registered in the furnace log.

In normal operational conditions the electrodes are slipped two-three times a day in furnaces smelting ferrosilicon and silicon chrome and every shift in those smelting all other grades of alloys. The time interval between slipping operations for self-baking electrodes should not be less than six hours. The slipping height varies from 100 to 250 mm, depending on the electrode diameter, the frequency of slipping and the technological features of the process.

Should there be a heavy evolution of gases following the slipping of an electrode, power load at the latter must be decreased to avoid breakage.

Electrode breakage may occur when an electrode is slipped into a moist spot under full load, due to rapid cooling when the furnace power is taken off, due to incorrect heating-up procedures following the furnace shut-down, due to poor ramming or poor quality of anode paste, etc.

Following a breakage an electrode should be slipped to a greater length and should be baked according to a special time-table.

The electrode-baking procedure following the breakage depends on the condition of the remaining working tip:

a) when an electrode breaks in a baked section, the broken part should be removed, the electrode should be slipped to the minimum length required for normal operation and baking should be started with power input at the electrode equal to 30% of the rated. The

magnitude of the current is then gradually increased at a rate which will make it possible for full furnace power input to be reached within six hours;

b) when an electrode breaks in the clamps and paste crumbles out, the furnace is shut down, the damaged electrode is set on the remaining electrode and its raw tip is pressed hard against it. The electrode is then slipped 700-1,000 mm and, when possible (in case the shell of the damaged section is intact), the damaged and the raw electrode are jointed by a steel band hoop (by means of acetylene welding), power is reapplied and baking is conducted by gradually increasing power load in the course of 12 to 16 hours.

Normal baking process is characterised by dark electrode shell, absence of red spots and swellings on the shell (openings are provided in the shell to permit the evolution of gases), evolution of short tongues of smoking flame through the openings. The reddening of the shell, outflow of paste and violent evolution of gases are signs of an overload; in this case power input should be decreased immediately. The disappearance of flames shows that the baking process is over.

Consumption of shells as well as that of the anode paste and electrodes is given in Table 6.

Table 6

**Anode Paste and Sheet Steel Consumption
in the Production of Certain Grades of Ferroalloys**

Alloy	Consumption in kg per ton of alloy	
	anode paste	sheet steel
Ferrosilicon 90%	80	4.5
Ferrosilicon 75%	40	2.0
Ferrosilicon 45%	24	1.2
Silicon chrome 50%	28	1.3
Silicon calcium	120	9.0
Ferrochrome BY and MY (flux)	15	1.2
Silicon manganese 14%	40	1.6
Ferromanganese CY	17	1.0
Carbon-bearing ferrochrome	40	1.6
Ferrotungsten	80	7.0

Part II

FERROALLOY TECHNOLOGY AND PRODUCTION METHODS

There is a considerable number of various ferroalloys and alloying metals used in the steel production. As far as the volume of production is concerned, the leading role among the ferroalloys belongs to the alloys of manganese and silicon.

Carbon ferromanganese and ferrosilicon with low silicon content are smelted in blast and electric furnaces. Ferrosilicons rich in silicon as well as refined grades of ferromanganese and silico-manganese are smelted in electric furnaces. Economy in ferroalloy production in blast furnaces is determined mainly by coke consumption per ton of alloy, while economy in their production in electric furnaces is determined by specific electric power consumption and the latter's cost.

The construction in the U.S.S.R. of large hydropower stations generating power at the cost not more than 0.2 kopeks per kWh, as well as of modern thermal power stations using cheap grades of fuels; with the power cost averaging 0.3 kopeks per kWh, permits of organizing a more economical production of ferroalloys in electric furnaces.

The necessity of using the poor manganese and carbonate ores available in the Soviet Union also promotes electrothermic ferroalloy production. For this reason it is planned gradually to decrease ferroalloy production in blast furnaces and to increase it in electric furnaces in the near future.

Chrome alloys are smelted mainly in electric furnaces, the exceptions being the special grades of ferrochrome and chrome-bearing alloys produced by the metallothermic method. Metallic chrome is also produced by this method, as well as by electrolysis. Ferrochrome decarburation in vacuum is being currently used for the production of extra low-carbon grades of ferrochrome. Various other grades of ferroalloys are smelted in electric furnaces or produced by the metallothermic method.

Pure alloying metals are produced by the electrolytical method. In the production of metallic manganese, this method makes it possible to use poor manganese ores.

Chapter 1

PRODUCTION OF SILICON ALLOYS

Designation and Composition of Silicon Alloys

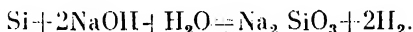
Silicon is a strong deoxidiser; this property determines basically its use in metallurgy. It is used as a deoxidiser in the production of all kind of steel (with the exception of rimming open-hearth steel) and sometimes as an alloying element.

Steel usually contains up to 0.35% Si; steel with a higher silicon content is classified as silicon alloy steel.

An addition of about 1.3-2.0% Si to steel increases its yield point and resilience, relative elongation, reduction in area, and impact strength. This is the so-called structural steel. Steel with 2.5-4.3% of silicon is characterised by low remanent magnetism and low active losses. It goes for the manufacture of transformer and dynamo cores.

High-percentage ferrosilicon in the form of powder is used as a slag deoxidiser; large amounts of ferrosilicon are used for the reduction of various oxides in ferroalloy production; for example, in the production of ferromolybdenum, ferrovandium, etc.

Ferrosilicon is a component of mixtures used for heating ingot heads (lunkerite) and for manufacturing hydrogen by the following reaction:



Commercially pure silicon is used in non-ferrous metallurgy for the manufacture of silicon bronzes, etc.

Chemical composition of all ferrosilicons used in metallurgy is standardised by state standards and technical specifications (state standards for blast-furnace ferrosilicon ГОСТ 5163-49, for electrothermic ferrosilicon ГОСТ 1415-49, ЧМТФ 21-58 (Table 7).

ГОСТ 1415-49 ferrosilicon is delivered in crushed lumps less than 25 kg in weight. The amount of fines on a 20×20 mm screen should not exceed 15% of the total weight for the Cn90 and Cn75 grades and 25% for the Cn45 grade.

No sharply defined non-metallic inclusions are tolerated.

Table 7

**Composition of Commercial Grades of Ferrosilicon
and Crystalline Silicon**

Grade	Chemical composition, %								
	Si	Fe	Mn	Al	Cr	P	Ca	S	Sum of impurities
	Maximum								
<i>Blast-Furnace Ferrosilicon (FOCT 5163-49)</i>									
Cu10	9.00-13.00	—	3.00	—	—	0.20	—	0.04	—
Cu15	13.1 and over	—	3.00	—	—	0.20	—	0.04	—
<i>Electrothermic Ferrosilicon (FOCT 1415-49 and UMTV No. 21-58)</i>									
Cu90	87-95	—	0.5	—	0.2	0.04	—	0.04	—
Cu75	74-80	—	0.7	—	0.5	0.05	—	0.04	—
Cu45	40-47	—	0.8	—	0.5	0.05	—	0.04	—
Cu25	22.1-26	—	2.0	—	—	0.10	—	0.04	—
Cu18	17-22	—	2.0	—	—	0.10	—	0.04	—
<i>Crystalline Silicon (FOCT 2169-43)</i>									
	Above								
Kp0	99.0	0.5	—	0.5	—	—	0.5	—	1.0
Kp1	98.0	0.7	—	0.8	—	—	0.5	—	2.0
•Kp2	97.0	1.0	—	1.2	—	—	0.8	—	3.0
Kp3	95.5	1.5	—	1.5	—	—	1.5	—	4.5

Raw Materials for Smelting Ferrosilicon

Silicon-bearing materials. There is a variety of silicon-bearing substances serving as raw materials for the manufacture of ferrosilicon. Silicon is second only to oxygen in occurrence in nature. It constitutes 27.6% of the earth's crust. Man knows more than 200 different varieties of silicon: quartz, rock crystal, amethyst, opal, onyx, cornelian, jasper, sand and many others. Moreover, thousands of minerals occur in nature, in which silicon combines with oxides of different elements to form silicates.

The cheapest minerals rich in silicon are used for the manufacture of ferrosilicon. These are the quartz, quartzite and chalcedony.

Quartz is a compact mineral with crystal structure, specific gravity of 2.59-2.65, hardness 7, mostly colourless, white, grey or reddish, depending on impurities. Quartz is a relatively costly mineral and is used for the manufacture of crystalline silicon. Its SiO₂ content is 98% and over. Iron oxide in quartz should not exceed 0.3%. It should not have inclusions of pyrite and plates thicker than 1 mm.

Chalcedony is a thin-fibrous, sometimes pporous, mineral, variously coloured. Its behaviour in metallurgical processes is somewhat worse than that of quartz and quartzite as it contains a lesser amount of SiO_2 about 95%; its use is limited.

Quartzite is a rock composed of quartz grains, cemented by a substance mainly containing silicon. Quartzites are widely distributed and their reserves are inexhaustible. They are mined in open quarries, then crushed and screened in crushing and sizing mills.

Quartzite used in the production of ferroalloys should meet the following requirements:

- 1) silicon content should be minimum 96%, desirably 97-99%;
- 2) amount of slag-forming impurities (alumina, magnesium oxide, calcium oxide) should be the minimum possible;
- 3) P_2O_5 content should not exceed 0.02%;
- 4) it should have no clay inclusions;
- 5) its moisture absorption should not exceed 5%;
- 6) it should not lose its mechanical strength when crushed or heated.

Carbon-bearing quartzite (of the Chugunash type with 0.5-0.7% C) is not suitable for metallurgical smelting as it cracks when heated and thus upsets the gas permeability of the burden.

Chemical composition of quartzites in various deposits is given in Table 8.

Table 8

Chemical Composition of Quartzites

Deposit	Chemical composition, %							
	SiO_2	Fe_2O_3	Al_2O_3	CaO	MgO	TiO_2	P_2O_5	C
Bakal	97.20	0.73	1.15	0.58	0.20	0.07	—	—
Olgino	97.90	0.32	1.14	0.21	0.14	—	—	—
Tarasovo	96.08	0.84	0.22	0.72	—	—	—	—
Chugunash	96.00	2.24	1.12	0.70	0.17	—	0.011	0.5
Antonovka	98.16	0.23	0.53	0.60	0.05	—	0.014	—
Pervouralsk	98.25	0.41	0.24	0.70	—	0.14	—	—

Reducer. Various carbon-bearing materials may be used as reducing agents in ferrosilicon production: charcoal, petroleum and pitch cokes, metallurgical coke, and coal.

The reducer should meet the following main requirements: minimum ash content (chemical composition of ash is also of importance), large reactive surface, low volatile matter content, high electric resistance and cheapness. A sufficient mechanical strength is also required.

Charcoal is an excellent reducing agent. It contains very little ash (usually 1-2%), possesses high reactivity and considerable electric resistance. The presence of charcoal in the burden reduces the latter's sintering.

The usual composition of charcoal is as follows: 13% moisture, 18% volatile matter, 68% C and 1% ash. However, because of its high cost (20-50 rubles a ton), charcoal is used only for the production of high-silicon alloys: 90% ferrosilicon and crystalline silicon.

Petroleum and pitch cokes, containing negligible amounts of ash (less than 1-2%) and volatile matter without undesirable impurities and possessing high porosity and reactivity, are excellent reducing agents. However, because of their high cost these reducers are used only for the production of crystalline silicon and ferrotungsten.

Coals have high electrical resistance; some grades contain little ash, as well as negligible quantities of volatile matter and undesirable impurities. But they crack when heated and this restricts their use. Substitution of coal for 20-30% of fine coke decreases the sintering of the charge and permits a deeper positioning of electrodes. The question of the expediency of using coal as a reducing agent for ferrosilicon production cannot as yet be considered solved.

Metallurgical fine coke (nut size) is the basic reducer in ferrosilicon production processes. Metallurgical fine coke is a sifting waste of metallurgical coke production. It contains considerably more ash than coke and has a high, widely variable, moisture content. Fine coke obtained from various coals conducts itself in different ways in metallurgical smelting. For example, Gubakha coke has a higher ash content and porosity, possesses higher electrical resistance and makes for deeper positioning of electrodes. The chemical composition of coke and fine coke is given in Table 9 and ash content in Table 10.

Composition of Coke and Fine Coke

Table 9

Reducer	Composition, %						Size, %	
	Moisture	Ash	Volatiles	P	S	Fixed C	25-40 mm	Fines
Donets coke	5.0	9-10	1.90	—	1.4-1.5	81.60-82.70	—	—
Kuznetsk coke	5.0	11-12	1.00	—	0.4-0.5	81.50-82	—	—
MMK fine coke	25.2	12.00	1.50	—	—	86.50	2.0	6.0
Kuznetsk fine coke	13.2	10.47	1.20	—	0.49	87.84	—	7.5
ЧМЗ fine coke	14.5	10.50	1.40	0.058	0.49	87.55	2.2	4.2
Gubakha fine coke	17.5	16.35	1.78	—	—	81.87	—	6.5

Table 10

Approximate Composition of Fine Coke Ash

Reducer	Chemical composition, %						
	SiO ₂	Al ₂ O ₃	Fe ₂ O ₃	CaO	MgO	P	FeO
4M3 fine coke	48.52	29.78	—	6.50	2.20	0.58	9.67
Gubakha fine coke	44.32	36.92	13.19	2.60	1.00	0.076	—
Novokuznetsk fine coke	45.16	28.93	9.77	6.30	2.00	0.33	—
Kemerovo fine coke	48.52	28.17	11.61	6.60	2.20	0.37	—

Experiments were recently conducted with a view of using the waste of electrode-graphitising process as a reducing agent. The chemical composition of these wastes is as follows:

SiC	SiO ₂	C
23-40%	40-52%	20-25%
16-29%	12-25%	50-60%

The following results, in terms of specific power consumption, were obtained from the 30 % of waste contained in the reducing mixture: for Cn45—4460 kWh per ton; Cn75—8260 kWh per ton; CnXp50—5220 kWh per ton. Production costs were at the same time lowered as a result of economy on power and raw-material consumption.

Timber industry waste (saw dust, chips, cuttings) is now used as a reducing agent in the U.S.S.R. and elsewhere.

The use of these tailings offers the following advantages:

1) it ensures satisfactory gas permeability of the burden and prevents charge crusting;

2) it lowers the specific electrical conductivity of the charge and thus makes for deeper positioning of electrodes.

Electrode penetration may be regulated to a certain degree by varying the amount of timber waste in the burden mixture.

Iron-bearing materials. Iron may be introduced into the charge in the form of iron ore, scales or filings. The use of iron raises power costs (to reduce iron oxides and heat the slag) and requires an additional amount of reducer.

The use of scales somewhat improves furnace-top operation and ensures fuller evacuation of slag, but entails additional power and reducer consumption to reduce iron oxides.

It is common practice in ferrosilicon production technology to introduce iron into the mixture in the form of cuttings. The cuttings should be free from rust, as the latter is a source of increased hydrogen content in ferrosilicon. Cuttings of alloy steel should not be used, since all alloying additions (chrome, nickel, manganese, etc.) go into the alloy in full, while their content in the latter is strictly limited by the state standards. Use of cast-iron cuttings is also

prohibited because the phosphorus they contain almost completely passes into the alloy. The presence of sulphur is not dangerous as it forms volatile compounds with silicon and evaporates in the smelting process. The finer the cuttings the easier it is to handle them and the more uniform is their distribution in the burden; therefore, the use of large and entangled cuttings is not recommended. Cuttings should be free from impurities, especially non-ferrous; the use of greasy cuttings is not advisable as they hamper furnace-top operation.

The iron content in cuttings comes to about 90%.

Presmelting Preparation of Burden Materials

Presmelting preparation of burden materials is an important operation which often predetermines performance results in ferrosilicon smelting.

Quartz and quartzite which are used for smelting ferrosilicon are crushed and sieved. For smelting 45% ferrosilicon, quartzite should be broken into lumps 35 to 70 mm in size when machine charging is available; the use of 70 to 120 mm lumps is also permissible. Fines should be sieved through a 20- to 30-mm sieve. Flushing of quartzite reduces the amount of slag-forming matter and thus improves furnace operation. Quartzite should be as uniform in size as possible, for this provides, all other conditions being equal, better gas permeability of the charge. The presence of fines in the mixture is undesirable not only because it decreases its gas permeability, but also because the fines contain a higher percentage of impurities, especially alumina (20% and over).

Quartz, used in the mixture for the production of crystalline silicon, is separated from fine particles (which form in the course of conveying and charging operations) with the aid of a 15-mm sieve. Quartz separated from the fines is crushed and conveyed to the furnace without a second sieving. The lumps of quartz fed into the furnace should not exceed 50 mm.

Fine coke is sieved to separate fines, then crushed, and screened again to separate large particles, as well as fines, which form in the crushing process. The size of fine-coke lumps considerably affects the operation of the furnace. The size of fine-coke lumps should be chosen in accordance with furnace rating and its secondary voltage; for example, it is best to have fine coke in 10- to 20-mm lumps for furnaces of 7,000 to 10,000 kVA with secondary voltage of 140 to 160 V. When higher voltages are used, the size of fine-coke lumps should be reduced. But whatever conditions, lumps of less than 5 mm are not suitable for ferrosilicon production and should be sifted out.

Table 11 shows the relation of specific power consumption in ferrosilicon smelting to the amount of coke fines in the mixture.

In all cases the charge particles, especially that of the coke, should be of the same size. Deviation in the size of coke in any direction leads to serious disturbances in the operation of ferrosilicon furnaces.

Prior to charging, iron cuttings should be crushed in milling machines (maximum coil size 50 mm).

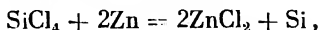
Table 11

Relationship Between Power Consumption in Ferrosilicon Smelting and the Amount of Coke Fines in the Mixture

Alloy	Characteristics of coke introduced into the charge	Furnace campaign	Specific power consumption, kWh per ton
Cu45	Coke siftings below 5 mm amounting to 11% of the reducer weight; the rest — 5 to 20 mm nuts	From April 17 to March 12, 1955	5,080
Cu45	Nuts 5 to 20 mm	January-March 1955	5,054
Cu75	Siftings below 8 mm amounting to 12% of the reducer weight; the rest—8 to 25 mm nuts	August 18 to 22, 1955	9,130
Cu75	Nuts 8 to 25 mm	August 1 to 17 and August 30-31, 1955	8,960
Cu75	Fine coke 0-20 mm without sifting	March 31 to April 18, 1956	9,120
Cu75	Fine coke 8 to 20 mm	March 1-30 and April 19-30, 1956	8,945

Physicochemical Properties of Silicon, Iron and Ferrosilicon

Silicon is the fourteenth element of the Mendeleyev Periodic System. Initially, in its pure form, it was obtained in 1811 by Gay-Lussac and Thénard by passing silicon fluoride over heated potassium; its properties, however, were described only in 1823 by Berzelius who named the element silicon (from Latin *silix*—stone) and gave the chemical symbol Si. In crystalline form silicon was obtained for the first time in 1855 by Deville. Pure silicon obtained by the method of Academician Beketov:



is a brown highly hygroscopic powder. Macrocrystalline silicon is obtained when silicon is melted and cooled; its colour is grey steel with a metallic lustre, it is hard and brittle. The physicochemical properties of silicon are as follows:

Atomic weight	28.06
Specific gravity	2.4
Valence	2 and 4
Melting point, °C	1414
Boiling point, °C	2287
Heat of fusion, cal/mol	9,470
Specific heat capacity at 25°C, cal/°C	0.1762
Heat conductivity at 20°C, cal cm/cm ² sec °C	0.20
Electric resistance at 20°C, microhm cm ² /cm	0.000085

Chemically silicon is a typical metalloid.

Iron (Ferrum), Fe is an element of the VIII group of the Mendeleev Periodic System. Its ordinal number is 26, its atomic weight 55.85. Iron is a lustrous, silvery white, resilient and malleable metal.

Specific gravity	7.86
Melting point, °C	1539
Boiling point, °C	3080
Heat of fusion, cal/mol	3,070
Specific heat capacity at 25°C, cal/°C	0.108
Tensile strength, kg/mm ²	30
Extension elongation, %	40

Maximum valence of iron is +6, but the usual valence is +2 and +3. At higher valences iron shows signs of acidity.

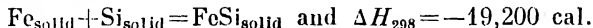
Iron corrodes readily and is easily alloyed with metals, as well as with non-metals.

Silicon and iron alloys. Silicon and iron can be alloyed in any proportion. A silicon-iron equilibrium diagram is given in Fig. 67. A number of chemical compounds exists in this system:

ε -FeSi (33.3% of Si); η -Fe₃Si₂ (25.1% of Si); ζ -Fe₂Si₃ (55.68% of Si).

Of the above-mentioned compounds only FeSi fuses (at 1410°C) without decomposition; the rest disintegrate at high temperatures.

The heat of formation of FeSi has been determined experimentally for the reaction



For the melted substances the heat of formation is $\Delta H_{1878} = -29,000$ cal at 1878°K.

The high value of the heat of formation, as well as the form of the curve in the maximum zone of the equilibrium diagram, show that FeSi is stable.

There are three eutectics in the system: the first eutectic α - ε (20% Si) with a melting point of 1195°C, the second ε - ζ (51% Si) with a melting point of 1212°C and the third ζ -Si (59% Si) with a melting point of 1207°C.

The diagram shows that the melting point of standard grades of ferrosilicon does not exceed 1370°C; a 40-47% silicon alloy melts at 1370-1260°C, while a 74-80% silicon alloy melts at 1320-1340°C.

The specific gravity of iron-silicon alloys varies with their silicon content. This fact is made use of in smelting practice to determine the silicon content in the alloy. The method, however, is not very precise, for commercial alloys comprise various other elements (besides silicon and iron) which affect its specific gravity. But, as smelting practice shows, this method is quite acceptable. Variation in the specific gravity of the alloy in relation to its silicon content is shown below:

Silicon content, %	40	45	50	55	60	65	70	75	80	85	90
Specific gravity of alloys	5.61	5.15	4.75	4.37	4.00	3.76	3.51	3.27	3.03	2.78	2.55

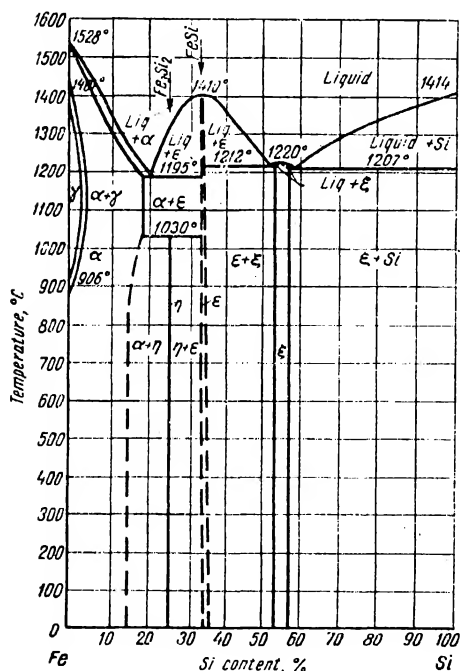


Fig. 67. Equilibrium diagram of iron-silicon system

Some grades of silicon readily disintegrate when stored in moist atmosphere; the pulverisation of ferrosilicon is accompanied by the evolution of gases, mainly hydrogen, phosphorus hydrogen PH_3 and arsenious hydrogen AsH_3 . The last two are very poisonous. Their presence is readily discerned by a strong garlic odour. Repeated cases of poisoning, especially aboard vessels carrying ferrosilicon, as well as explosions and fires caused by the formation of self-igniting phosphorus hydrogen P_2H_4 , have prompted investigations into the silicon powdering effect.

The work was initially undertaken by N. S. Kurakov and G. G. Urazov and the results were published in 1914 in the *Mining Journal*. The authors found that not all grades of ferrosilicon were subject to pulverisation, but only those which contain from 33.3 to 75% Si, and especially alloys with 50 to 60% Si. Because of this, Soviet state standards for ferrosilicon make provision

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only for those alloys (45, 75 and 90% silicon) which show a minimum tendency to disintegrate. However, ferrosilicon has a strong tendency to segregate as a result of which the difference in silicon content between the top and the bottom of the ingot may amount to 20 and more per cent. (Segregation is uneven distribution in height of an ingot of various elements which differ from each other as to specific gravities in an alloy; the greater the difference in specific gravities of the alloy components and the more the time interval of ingot solidification, the more pronounced segregation is.)

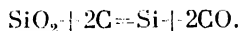
Laboratory research has established that pure iron-silicon alloys do not disintegrate in the air, notwithstanding the quantity of silicon they contain. Alloys containing 0.03-0.04% P and alloys with up to 3% of Al do not disintegrate either; simultaneous presence of 0.03-0.04% of P and 3% of Al causes its rapid disintegration in moist air. Rapid cooling prevents alloy disintegration; slow cooling increases segregation as a result of which the bottom of the ingot is impoverished in silicon (75% ferrosilicon) or the top of the ingot is enriched in silicon (45% ferrosilicon); conditions favouring powdering are created in these zones. Therefore, alloys destined for long storage are teemed, in ingots maximum 100 mm thick, in cast-iron moulds to speed up cooling.

Physicochemical Principles of Ferrosilicon Smelting

Silica, introduced into the mixture in the form of quartz or quartzite, is reduced in the process of ferrosilicon and crystalline silicon smelting.

Silica is reduced by solid carbon, for there is little likelihood of its reduction by carbon monoxide ($\text{SiO}_2 + 2\text{CO} = \text{Si} + 2\text{CO}_2$) in the absence of fixed carbon. Calculations show that at 2000°C the gas mixture, containing as little as thousandths of a per cent of CO_2 , is an oxidising agent as far as silicon is concerned.

The mechanism of silica reduction by carbon has not been studied sufficiently, but the summary reaction of the reduction process may be presented as follows:



In silicon-alloy smelting practice this reaction is used for various calculations.

The latter show that the reduction of SiO_2 by carbon begins at about 1500°C, while at 1800°C the reaction is practically complete.

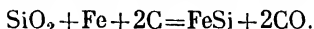
A further rise of temperature is not worthwhile, for the intense reduction of silicon is attended by considerable evaporation.

There are a number of references in literature as to the favourable effect which iron has upon the reduction of silica. It has been proved

by experiments that in the presence of iron some insignificant reduction of silica is possible even at 1200°C.

Analysing the silica reduction process in an electric furnace, Prof. Maximenko points out that iron promotes the sedimentation of silicon, thus eliminating the latter from the reactive zone and favouring the reduction of silica.

The reduction of silica by carbon in the presence of a sufficient amount of iron may proceed according to the following reaction:

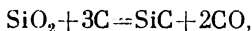


Power consumption per unit of silicon reduced drops along with the decrease in silicon content in the ferrosilicon.

Assuming that power consumption for the melt-down of one ton of iron, contained in ferrosilicon, averages 500 kWh, power consumption per kg of silicon will amount to approximately (in kWh per kg):

For 90% ferrosilicon	13.0	For 45% ferrosilicon	10.0
" 75% "	11.8	" 18% "	9.2

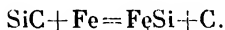
The following reaction may develop:



when an excessive amount of reducer is introduced into the furnace with the mixture.

Theoretical temperature for the outset of this reaction is 1530°C (1257°C), i.e., lower than the initial temperature of reaction of formation for Cr45 and Cr75. This is corroborated by the formation of carborundum in furnace-top cold spots during smelting ferrosilicon in a furnace, as well as by the rapid decrease in furnace volume due to the appearance of carborundum in furnaces smelting crystalline silicon when the furnace runs cold.

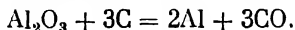
In ferrosilicon smelting there are no carborundum accretions in the furnace, since carborundum is destroyed by iron according to the reaction



This reaction finds application in ferrosilicon production when the carborundum-bearing waste of the electrode-manufacturing process is used as a raw material.

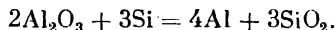
Quartzite and reducer ash are also reduced in the furnace along with silica.

Aluminium may be reduced by carbon according to the reaction:

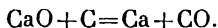


This reaction is possible only in the presence of elements capable of dissolving aluminium; in the above-mentioned case such elements are iron and silicon. The feasibility of this reaction is cor-

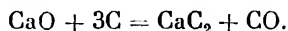
roborated by the presence of 1.5-2% aluminium in commercial grades of ferrosilicon. It should be pointed out that when the concentration of silicon is high the reduction of aluminium may proceed as follows:



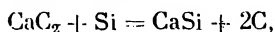
The reduction of CaO by carbon in the presence of solvents of calcium—silicon and iron—proceeds according to the reaction



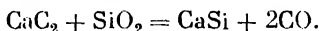
In the reduction of CaO an intermediate product—calcium carbide—may be obtained:



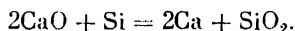
This reaction proceeds more readily than the preceding one but has no major significance, for CaC_2 decomposes in the presence of silicon according to the reaction



more probable is the reaction

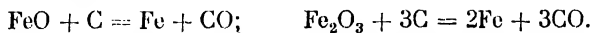


The following reaction may take place when there is a considerable excess of silicon:



Reactions of aluminium and calcium reduction begin at high temperatures and proceed with difficulty and are not completed.

The reduction of iron oxides contained in the mixture proceeds according to the reaction:



The theoretical temperature for the beginning of the first reaction is 712°C ; therefore, the reduction of iron oxides at temperatures of ferrosilicon-smelting processes is practically complete.

Almost all the phosphorus contained in the burden passes into the alloy; pure (as regards phosphorus) charge materials should be used.

The presence of sulphur in the charge is not dangerous, since it forms volatile compounds SiS and SiS_2 , and they evaporate.

Formation of Slag

Production of ferrosilicon is a slag-free process; however, the smelting of alloys is always attended by the formation of a certain amount of slag, this indicating that the slag is formed in the furnace.

Slag formation is caused by impurities in the raw materials—alu-

mina, calcium oxide, magnesium oxide, etc.—which cannot be reduced completely in the prevailing physicochemical conditions.

As alumina accounts for the greater part of the impurities, it obviously is the chief slag constituent. Slag formation can be represented as the slagging of a certain amount of silica by alumina and basic oxides. In ferrosilicon production, therefore, slag is aluminous.

When there is a lack of reducer, slag is enriched in silica as well as in carborundum, though that destroys the protective lining.

Slag analyses are given in Table 12.

Table 12

Slag Composition

Slag component	Amount, %				
SiO ₂	14.22	13.76	26.02	42.03	46.48
SiC	5.76	9.6	7.20	6.0	8.6
FeO	2.84	2.06	0.23	2.02	2.02
CaO	5.20	7.30	9.0	9.44	9.94
Al ₂ O ₃	10.55	15.03	21.45	40.50	36.04
BaO	3.26	4.86	8.07	—	—
MgO	0.58	0.61	0.43	1.81	1.61
Fe	29.64	24.04	—	—	—
Si	27.86	23.15	11.84	—	—

Slag composition varies widely as is evident from the table above, but all slags are viscous and hard to fuse. Their melting point is about 1700°C; viscosity at this temperatures equals 30-35 poises.

Slag fall is usually negligible (20 to 60 kg per ton of alloy) and is a function of the purity of raw materials and the grade of the ferrosilicon smelted. In contaminated mixtures the amount of slag formed increases to 100 kg per ton of alloy, thus sharply decreasing the furnace productivity and increasing power consumption per ton of alloy produced.

Precise estimation of slag fall in ferrosilicon smelting is rather difficult as the slag's specific weight is almost equal to that of the alloy and it is entangled in the mass of the metal obtained, especially when crystalline silicon is smelted.

Moreover, due to its high viscosity, some slag remains in the furnace and causes accretions which decrease the furnace yield and increase specific power consumption.

Slag formation in ferrosilicon smelting is mainly prevented by the strictly sufficient supply of reducing agent and the use of pure raw materials. Slag formed in the furnace should be removed completely after each heat. In some smelters accumulations of slag are removed by an addition of lime into the furnace hearth. This dilutes the slag and permits its tapping.

Charge Calculations for the Production of 45% Ferrosilicon

The charge is calculated per 100 kg of quartzite.

Let us assume that the raw-material assay is as follows:

<i>Quartzite:</i> 98.0% SiO_2	<i>Fine coke (dry):</i> 85% C
1.0% Al_2O_3	13% ash
0.6% Fe_2O_3	2% volatile matter
0.4% CaO	

<i>Coke ash:</i> 50% SiO_2	<i>Iron chips:</i> 94% Fe
25% Al_2O_3	6% losses when calcinated
20% Fe_2O_3	
5% CaO	

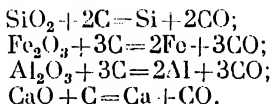
Assume that the distribution of oxides in the smelting process is as follows, %:

	SiO_2	Fe_2O_3	Al_2O_3	CaO
Oxides reduced	98.0	99.0	85.0	50.0
Oxides slagged	2.0	1.0	15.0	50.0

Assume that the distribution of the reduced elements is as follows, %:

	Si	Fe	Al	Ca
Reverted to metal	94	99	90	70
Losses	6	1	10	30

Deoxidising reactions:



The amount of oxides reduced and the oxygen evolved per 100 kg of quartzite will be equal to, kg:

$\text{SiO}_2 \dots 98.0 \times 0.98 = 96.04;$	$\frac{96.04 \times 32}{60} = 51.22$
$\text{Fe}_2\text{O}_3 \dots 0.6 \times 0.99 = 0.59;$	$\frac{0.59 \times 48}{160} = 0.18$
$\text{Al}_2\text{O}_3 \dots 1.0 \times 0.85 = 0.85;$	$\frac{0.85 \times 48}{102} = 0.40$
$\text{CaO} \dots 0.40 \times 0.50 = 0.20;$	$\frac{0.20 \times 16}{56} = 0.06$
Sum of $\text{O}_2 = 51.86$	

Amount of carbon necessary to combine with 51.86 kg of oxygen to give carbon monoxide:

$$\frac{51.86 \times 12}{16} = 38.89 \text{ kg.}$$

A certain amount of carbon introduced with the fine coke is consumed for the reduction of oxides contained in coke ash.

The amount of oxygen evolving in ash deoxidising is given in Table 13.

Table 13

Oxide	Reduced from 100 tons of fine coke, kg	Amount of oxygen evolving in reduction process, kg
SiO ₂	$13 \times 0.5 \times 0.98 = 6.37$	$6.37 \times \frac{32}{60} = 3.40$
Al ₂ O ₃	$13 \times 0.25 \times 0.85 = 2.76$	$2.76 \times \frac{48}{102} = 1.30$
Fe ₂ O ₃	$13 \times 0.20 \times 0.99 = 2.57$	$2.57 \times \frac{48}{160} = 0.77$
CaO	$13 \times 0.05 \times 0.50 = 0.33$	$0.33 \times \frac{16}{50} = 0.11$
		Total 5.58

Amount of carbon necessary for the reduction of coke oxides:

$$\frac{5.58 \times 12}{16} = 4.19 \text{ kg.}$$

Distribution of 85 kg of carbon contained in 100 kg of fine coke is as follows:

for the reduction of ash oxides 4.19 kg;

for the reduction of quartzite oxides $85 - 4.19 = 80.81$ kg, or 80.81%.

Amount of carbon necessary for the reduction of 100 kg of quartzite is 38.89 kg, or $\frac{38.89}{0.8081} = 48.13$ kg of fine coke.

Fine coke burns up partly at the furnace-top; practice shows that approximately 10% of fine coke is lost in this manner; therefore, the amount of fine coke necessary is: $\frac{48.13}{0.90} = 53.48$ kg.

Elements reduced in 100 kg of quartzite and 53.48 kg of fine coke are given in Table 14.

Table 14

Element	From quartzite	From fine-coke ash	Total, kg
Si	$96.04 - 51.22 = 44.82$	$53.48 \times 0.13 \times 0.98 \times 0.5 \times \frac{28}{60} = 1.59$	46.41
Fe	$0.59 - 0.18 = 0.41$	$53.48 \times 0.13 \times 0.99 \times 0.20 \times \frac{112}{160} = 0.96$	1.37
Al	$0.85 - 0.40 = 0.45$	$53.48 \times 0.13 \times 0.85 \times 0.25 \times \frac{54}{102} = 0.78$	1.23
Ca	$0.20 - 0.06 = 0.14$	$53.48 \times 0.13 \times 0.50 \times 0.05 \times \frac{40}{56} = 0.12$	0.26

Distribution of reduced elements is given in Table 15.

Table 15

Element	Reverts to alloy, kg	Stack losses, kg
Si	$46.41 \times 0.94 = 43.63$	$46.41 - 43.63 = 2.78$
Fe	$1.37 \times 0.99 = 1.36$	$1.37 - 1.36 = 0.01$
Al	$1.23 \times 0.90 = 1.10$	$1.23 - 1.10 = 0.13$
Ca	$0.26 \times 0.70 = 0.18$	$0.26 - 0.18 = 0.08$
Total	46.27	Total 3.00

When 45% ferrosilicon is smelted, 43.63 kg of silicon represent 45% of the total alloy weight. Therefore, the total weight of the alloy is $\frac{43.63}{0.45} = 96.9$ kg.

Amount of iron necessary is

$$96.96 - 46.27 = 50.69 \text{ kg}$$

$$\text{or } \frac{50.69}{0.94} = 53.93 \text{ kg of iron chips.}$$

The above-mentioned charge calculations are somewhat simplified. A number of charge components (carbon and iron of the electrodes, etc.) have been omitted. Thus electrode consumption per ton of 45% ferrosilicon is equal to 25 kg. Half of the consumed electrode paste goes for the reduction of oxides. Consequently, the amount of coke necessary is less than indicated in the calculations.

As 96.96 kg of alloy are obtained from 100 kg of quartzite, the amount of coke necessary should be decreased by

$$\frac{25}{2} \times \frac{96.96}{1,000} = 1.2 \text{ kg.}$$

The composition of a standard charge per 300 kg of quartzite is, kg:

Quartzite	300
Fine coke	$(53.48 - 1.2) \times 3 = 157$
Iron chips	$53.93 \times 3 = 162$

Practically, the weight of the fine coke in the charge varies with the ash and moisture content. The weight of the iron chips also varies, because of fluctuations in the iron content.

Charge calculations for the production of other silicon alloys are similar to those above, but in considering high-percentage alloys one should take into account the increased stack losses of reduced elements.

Technology of Ferrosilicon Production

Ferrosilicon is smelted in one- or three-phase electric arc furnaces usually of more than 7,000 kVA rating. The furnaces mostly used for the production of ferrosilicon are circular three-phase furnaces with self-baking continuous electrodes.

Furnaces with rotating baths are coming into use of late.

Practice shows that the use of furnaces with rotating baths reduces power consumption by 3-4%, correspondingly raises productivity, saves raw materials and drastically diminishes the need for labour-consuming furnace-top operation (due to the latter's soft and smooth run). No poking is required on the furnace-top. This is of utmost importance for the closed-top furnaces.

The use of closed-top furnaces for ferrosilicon smelting is important for improving the over-all performance of the process, for it allows to use furnace gases with a calorific capacity of 2,500 Kcal per cubic metre. As it stands now, the problem requires further investigating and designing.

Electrical parameters of the ferrosilicon-smelting process usually remain constant; voltage on the low side is 130-190 V at 35,000-45,000 A. In efficient furnaces of 10,000 kVA rating the I_{line} to E_{line} ratio equals approximately 270 (where I stands for current and E for voltage).

Furnace electrical operating conditions require an optimum diameter of electrode circle, which depends on furnace power rating and its design specifications; for a given furnace a larger electrode circle is needed when smelting 45% ferrosilicon.

Ferrosilicon is smelted by a continuous process. Therefore, it is of utmost importance from the standpoint of a correct smelting procedure to provide normal conditions for furnace-top operation.

The normal run of the furnace-top, as well as of the smelting process as a whole, is characterised by the following characteristic features.

1. Uniform gassing along the whole furnace-top surface, absence of dark sintered areas, absence of local gas blow-outs, uniform descent of the charge along the electrode cones and in the centre of the furnace between the electrodes, no charge downsides (when the charge contains wood chips such downsides are unavoidable, but they are smooth and no bright, white flame shoots out from underneath the electrodes).

2. Electrode cones are normal, the charge level in the centre of the furnace is lower than that on the perimeter, the poking of the charge is easy everywhere.

3. Stable deep positioning of the electrodes: in 45% ferrosilicon smelting, electrode penetration is not less than 800 mm, while for

75% ferrosilicon and crystalline silicon the penetration is minimum 1,200 mm.

4. Liquid slag is regularly present at each tapping; a light yellow flame shoots out through the taphole at the close of tapping; the taphole is easily poked and readily plugged.

5. Power consumption is stable and the recording wattmeter draws a smooth curve.

6. Per hour charging time-table is observed strictly in accordance with actual power consumption.

Feeding is continuous, by small portions, as fast as the charge sinks; the hottest zones of the furnace are charged first* (inner zones of the electrode cones).

Excessive feeding raises the smelting zone and disturbs the heat exchange therein; insufficient feeding increases heat and silicon losses.

Charging is the most difficult operation in silicon alloys smelting; until recently this was done mainly by hand. The difficulty of charging three-phase ferrosilicon furnaces lies in the irregular burden movement at different points of the furnace and consequently in the need of selective charging. The diverse specific weights and sizes of the charge components cause segregation in the mixture; the charge is composed of alternate layers of quartzite and fine coke.

At present, open-top three-phase furnaces in Soviet smelters are charged by Pluiko machines which operate on the throw principle. Machine-charging is satisfactory, especially in the case of low-percentage alloys. The drawbacks of the machines are their bulkiness and complexity of gearing.

Charging through furnace bins and chutes, located around the electrodes, will undoubtedly be widely applied as furnace rating rises and the use of rotating bath furnaces becomes more common. In closed-top furnace operation feeding may be ensured either by screw feeders (which exclude segregation) or by chutes around the electrodes.

The number of charges per shift is determined by furnace power consumption and specific power consumption per charge. It is 1,400 kWh per charge for 75% ferrosilicon (the charge contains 300 kg of quartzite). For crystalline silicon it amounts to 500 kWh per 100 kg of quartzite.

It is forbidden to charge unmixed materials or to draw components from a prepared mix.

Liquid silicon is quite volatile, i.e., silicon losses from evaporation are considerable. Furnace operators should conduct the process with a view of minimising silicon volatilisation and simultaneously increasing the amount of charge smelted to the maximum. This is achieved by deep and stable electrode positioning and uniform evolution of gases over the whole charge surface.

Electrode penetration is determined by the voltage drop between the electrode and the furnace bottom, by the magnitude of the electrode current and the electrical resistance of the mixture.

The electrical parameters of a furnace (electrode secondary—voltage and current) considerably affect furnace performance. Therefore, a correct choice of secondary voltage and the working current values for a ferrosilicon-smelting furnace is of utmost importance.

In practice, electrode penetration in the mixture is sometimes regulated by varying the electrical resistance of the furnace.

Furnace resistance is adjusted by increasing or decreasing the conductivity of the mixture either by changing its composition or the size of the lumps. An increase in the amount of carbon reductant in the mixture raises the latter's conductivity; an increase in the size of the coke used also raises conductivity; substitution of Gubakha coke for ordinary coke lowers the mixture's conductivity; so does the addition of wood tails.

The expansion of the electrode circle increases the electrical resistance of the furnace.

Concentration of power input in the electrical arcs speeds up deoxidation in the electrode crucibles as well as the melt-down of the mixture in the vicinity of the electrode. The portions of materials replacing the mixture consumed in the electrode crucibles descend by gravity from the top. The nearer the mixture is to the electrode the greater is its heat consumption and the greater its melt-down speed; the further away it is the less heat it consumes and the slower its descent is. The mixture situated further away from the electrodes is heated up mainly through heat conductance. The burden near the furnace walls is not heated up sufficiently to cause its smelting and the charge materials sinter into a monolith (protective lining).

A sort of crucible forms beneath each electrode (the hottest zone of the furnace) where most of the silicon reduction occurs.

The formation of electrode crucibles is explained by the uneven heating of the mixture and by the difference in smelting speeds in various sections of the furnace hearth. The mixture melts near an electrode; a certain distance away, it is either paste-like or half-sintered and forms the walls of the crucibles.

The colder mixture in the upper part of the crucible also sinters and roofs over the crucible. The walls and the roof of these crucibles melt down continuously and are replaced by new portions of the mixture. The crucible thus cannot be regarded as a frozen vessel. It is rather a high-temperature zone formed at the tip of the electrode. When the furnace runs hot the lower parts of the crucibles join to form a common crucible.

The rapid descent of the mixture near an electrode promotes the friability of the mixture column around it. Reduction reactions,

as well as the heaviest gas evolution, occur mainly around the electrodes. The temperature of the mixture and its gassing decrease, while the compactness of the mixture layers increases with their recession from the electrode. Gases forming near the arcs possess sensible heat and, passing through the mixture layers situated above, heat the latter.

When these upper layers of mixture are composed of readily reducible oxides (for example, iron oxides), the latter can be reduced by carbon monoxide. The flow of hot gases through a cold mixture causes the vapour of silicon and silicon monoxide to condense. If the gases pass through a wide area around the electrodes their speed is negligible and a large portion of vapour condenses in the mixture. The narrowing of the area of friable materials around the electrodes decreases the transfer of heat to the mixture and increases evaporation losses. When the zone of gas evolution around the electrode is narrowed down pressure in the furnace rises thus worsening the conditions for the reduction of silicon by carbon and creating local hot spots.

Poking at regular intervals is necessary in ferrosilicon smelting to provide uniform distribution of the escaping gases at the furnace-top, as well as to prevent sintering and the appearance of hot spots. Poking should be especially energetic in the areas of slow burden descent. Poking should be done to the maximum depth and the pole should be held almost vertically, as poking the uppermost mixture layers alone is useless.

In normal operating conditions, no poking is necessary in 45% ferrosilicon smelting. When separate sintered areas appear they should be poked by rammers; single hard crusts should be torn off and shoved towards the centre of the furnace-top, while the exposed areas should be covered with coke (or, analysis permitting, fine coke with iron chips). The use of wood tails enhances the permeability of the mixture.

Most of the troubles caused by uneven gassing at the furnace-top can be eliminated by using furnaces with rotating baths.

When the proportioning of the charge components is upset or incorrect technology is applied the run of the furnace deteriorates. These troubles may be classified as follows.

1. Lack of reductant in the mixture: electrode positioning becomes unstable; power input fluctuates widely (the recording wattmeter registers a band of dashes instead of a smooth curve; the larger the band the greater the load instability and, consequently, the greater the lack of reductant); long quartz threads drop from the electrodes; the crucibles contract; the mixture becomes strongly sintered; gas blow-outs appear at the top; the taphole emits gas as gas pressure in the furnace rises sharply, and the gases are made

white by burning silicon (rather by its oxide); consistent, viscous slag runs out of the taphole (tapping of slag is discontinued altogether when working with a lack of reductant for sustained periods of time).

These are the symptoms of the crusting of the furnace. The first sign of crusting, especially in a furnace smelting 45% ferrosilicon, is the appearance of sharply expressed crucible boundaries with white flames over them.

It should be remembered that the top of a furnace smelting 75% ferrosilicon, and all the more so of one smelting 90% crystalline ferrosilicon, always is somewhat more crusted than the one smelting 45% ferrosilicon.

To remedy the situation one should add a reductant to the mixture and charge more coke, and sharply activate furnace-top servicing, especially poking. Alloy analysis permitting, it is useful to add iron chips with fine coke to the sintered areas of the furnace.

Operating the taphole becomes difficult when one works with a lack of reductant for a prolonged period of time. Taphole plugging becomes either troublesome or totally impossible, and the melt infiltrates in any direction in the arch and even leaks through the furnace lining due to its softening at the front wall. At the first sign of taphole trouble electrode crucible servicing above the taphole should be activated and additional coke should be charged.

When taphole trouble is serious and the above-mentioned measures are of no avail, the following method should be applied. The mixture around the electrode should be melted down, power turned off and the coke in the crucible (some 150-200 kg of it) pressed down to the maximum. When this operation is well performed the coke is pressed down into the slag and no high positioning of electrodes is observed. But even if it is, that should not trouble the operator as the operation on the whole gives positive results (and practice confirms it). In certain difficult instances this operation should be repeated two or three times, but not more than once a shift.

In the smelting of crystalline silicon lack of reductant is discovered by the following symptoms: greater electrode consumption, evolution of great amounts of white fumes, ejection of gas from the taphole under great pressure, and instability of power input. This abnormality is eliminated by adding some reductant. Sometimes, when there is a lack of reductant, unreduced silica flows through the taphole, thus obstructing it and making it impossible to tap the metal. In this case quartz should be extracted with iron rods; quartz in the taphole should be heated beforehand by an electrical tapping apparatus.

2. Excess of reductant: electrode positioning becomes high, blow-outs spurt from underneath the electrodes (the furnace "roars"),

crucibles contract, the arcs operate noisily (characteristic buzz), the mixture collapses alongside the electrodes, there are no quartz threads around the electrodes, electrode power input is stable, slag is absent, metal fall is negligible; the metal is cold (the first stage of furnace "crusting", on the contrary, is noteworthy for its hot metal).

Sustained operation with excessive reductant in the mixture causes the freezing of the furnace-top, the slag crusting of the furnace and a sharp drop in productivity; the excess of reductant is usually easily discovered.

To remedy the situation the amount of reductant in the mixture should be reduced and one or two batches of mixture with a diminished amounts of reductant (the so-called "heavy" charges) should be charged; it is forbidden to charge pure quartz or a mixture with less than 100 kg of fine coke per charge portion. The furnace-top operation should be activated at the same time.

When crystalline silicon is smelted the mixture in the crucible does not sinter at all if there is an excess of reductant; it becomes porous and descends with frequent downsides. The colour of the flame at the furnace-top is dark-red. The taphole is "dry", i.e., poking reveals no signs of a liquid mass, and there are no splashes. This abnormality is eliminated by reducing the amount of fixed carbon in the mixture by 0.5-1.0 kg and by charging two or three batches of materials with a lack of reductant. Moreover, it is recommended to substitute charcoal for fine coke in these charge portions.

In smelting silicon, the excess of reductant in the charge for a prolonged period of time leads to the appearance of carborundum which should immediately be removed from the furnace. To this end the mixture near the electrodes should be smelted for 30 minutes and carborundum should then be extripated out of the electrode crucibles with iron rods. If this operation is not successful, carborundum should be shattered and the pieces thrown under the electrodes. A burden with a lack of reductant should be charged following this operation.

If there is considerable accumulation of carborundum the furnace should be heated for two to four hours without charging. After that power should be turned off, the furnace hearth should be cleared of carborundum with the aid of pneumatic picks and power reapplied for normal operation.

3. The operation with short electrodes is similar, in symptoms and results, to that with excessive reductant.

Hot flames shoot out from underneath the electrodes, the crucibles narrow down, arc operation is noisy, the mixture tumbles alongside the electrodes, no slag is tapped, metal fall is negligible and the alloy is cold.

The electrodes should be slipped and lengthened immediately.

4. The operation with excessively long, high-positioned electrodes increases their power losses; in case of deep penetration, the electrodes settle down in the slag, arcing is lost, there is no smelting in the furnace and a considerable amount of power is wasted.

Moreover, the operation with excessively long electrodes usually leads to the crusting of the furnace because of the operators' tendency to sink the electrodes in order to set the furnace right.

In this case, electrode slipping should be curtailed to remedy the situation.

In all cases of furnace trouble the following factors should be checked:

1) size of raw materials and their conformity to the standards, since operation with coarse coke is similar, by its external symptoms, to operation with excessive reductant;

2) accuracy of weighing equipment and proper proportioning of the mixture;

3) conformity of the weight of the reductant to its moisture content.

Moreover, the operators' attention should be focussed on the intensification of furnace-top servicing, since furnace trouble is often the result of insufficient, belated and incorrect technological operations at the furnace-top while the proportioning and quality of raw materials comply with requirements for normal operation.

Alloy tapping and teeming. Ferroalloy furnaces smelting ferrosilicon operate continuously while the alloy is tapped periodically, as it accumulates in the bath. There are from four to eight tappings per shift, depending on the grade of ferrosilicon and teeming methods. Frequent tappings are inadvisable, for they result in higher teeming and dressing losses; on the other hand, rare tappings are not recommended either, since accumulation of excessive amounts of alloys in the furnace hinders its operation. The alloy must be tapped after a set period of time, according to a rigid time-table. At the start of each shift the taphole block is fettled with anode paste and covered with a layer of sand. The paste should fill perfectly all the roughnesses of the taphole. This is achieved by smoothing the heated paste (it is heated up by heat evolving from the furnace) with a tamper. At the same time the taphole is shaped to a shallow runner.

When the taphole is insufficiently cleared of slag and the recess around the eye of the taphole is filled with it, the accretions are destroyed by means of electric tapping apparatus and removed.

When operating normally, the eye of the taphole is readily pierced by an iron rod; when this method fails, the taphole is opened by electric arc struck between the cone of the hole and an iron rod connected to the electrode of the electric tapping apparatus.

The taphole should be opened wide enough to permit full evacuation of slag from the furnace, but at the beginning of tapping the hole is made small to prevent a violent breakthrough of the alloy, which may cause accidents and contaminate the ingots (a strong jet washes away sand from the runner). In normal conditions, slag should be evacuated from the furnace at each tapping of alloy. Slag accumulation in the furnace causes a breakdown in the operation of the taphole, disturbs the normal technology of the process and leads to a general disturbance in furnace run.

The duration of alloy tapping is 15-25 minutes. During the process the taphole is cleaned from time to time with an iron rod to clear the opening and ensure the evacuation of slag. It is forbidden to poke the taphole with a wooden pole, for this hampers the evacuation of slag and endangers the personnel.

When plugging the taphole, special care should be taken to clear it thoroughly of alloy and slag. To this end the taphole is heated by an electric tapper. If the taphole is crusted with slag and alloy, it is very difficult to plug it and negligence in its cleaning often causes serious accidents.

After the tapping, the eye of the taphole is plugged to the maximum depth with a conical electrode paste plug (the plug is prepared from melted paste and sand in a special mould in which the paste cools and consolidates) or with a mixture of fireclay and electrode flour. If closed with a shallow plug, the taphole causes the front wall lining to heat up and leads to taphole trouble and to lining breakthroughs near the taphole. It must be pointed out that this is one of the most frequent causes of furnace hearth accidents in silicon-smelting practice.

Ferrosilicon is tapped directly into moulds (which are rolled in under the taphole) or into a ladle. Crystalline silicon is tapped into an all-cast iron mould which is rolled in under the taphole on a carriage.

Moulds for ferrosilicon may either be cast or welded and lined with firebrick. In both cases the mould is fettled with sand, and a carbon block is placed under the alloy jet.

The alloy is first tapped into a ladle and then teemed into massive cast-iron moulds with ingots approximately 70 mm thick (when pouring metal a piece of alloy from a preceding heat is placed under the jet) or into sectional cast-iron moulds fettled with crushed or granulated alloy. This method of teeming permits to obtain a pure ingot in a single operation without further dressing, crushing excepted.

Reversible moulds have been tried out, which made it possible to mechanise the teeming of metal (Fig. 68). When a granulated alloy is desired, it is granulated in a special installation. The

installation for granulation consists of a receiving spout with a launder to receive the metal jet from the ladle and to direct it to water pulverisers, "the cutters" which break the metal into grains, of a granulation tank ($3 \times 2 \times 6$ m) for cooling the granulated metal and of a basket which slides into the tank and serves to collect the granulated alloy.

Teeming of ferrosilicon into massive cast-iron moulds with ingots approximately 70 mm thick permits to obtain homogeneous ingots. It has been established that there is a noticeable segregation of silicon and aluminium when ferrosilicon is cooled slowly (less than 10°C per minute). Carbon content in the top of the ingot is higher than in the bottom by 12 to 17%; aluminium content is higher in the ingot bottom by 1%, hence the tendency of the ingot bottom to crumble, especially in 75% ferrosilicon ingots. Rapid cooling (faster than 60°C per minute) permits to drive down the segregation of silicon to 2% and that of aluminium practically to nil, thus sharply decreasing the trend of ferrosilicon to disintegrate.

Therefore, to diminish segregation, i.e., to obtain a stable, non-crumbling ferrosilicon the alloy should be cooled faster. This may be done by teeming the alloy into metallic moulds and in ingots not exceeding 70-100 mm in thickness.

When 45% ferrosilicon has been tapped, the slag crust should be removed before it freezes.

Ferrosilicon is kept in the moulds until it fully solidifies; the surface of the ingot should be dark-red (Cu45 grade) or grey (Cu75). The ingot is then transported by a crane to a special cooling platform in the pouring aisle. Since the strength of ferrosilicon alloys drops with temperature causing cracks, ingot dumping should not be delayed. The duration of the ingot cooling operation depends on the type of the mould, its fettling and the alloy weight.

Granulated alloy is unloaded from the basket into a bin with holes for the evacuation of water; the alloy is held in the bin for at least 20 minutes and is then weighed.

Ingot is broken to fragments of desired weight by hand or by pneumatic hammers. Sand on the bottom surface of ferrosilicon ingots is cleaned off with a metallic brush, while the slag is hammered off.

Cleaned alloy is loaded into bins for weighing and handling.

As we have mentioned above, it is possible to obtain ingots without such dressing procedures, provided a granulated or finely crushed alloy is used for the fettling of moulds.

When the alloy is tapped out into massive reversible moulds some alloy fragments from a preceding heat are put under the jet to protect the mould against erosion. This permits to deliver the alloy to customers without additional dressing.

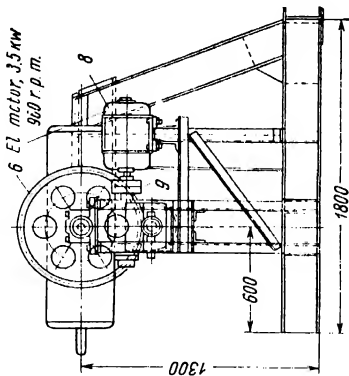
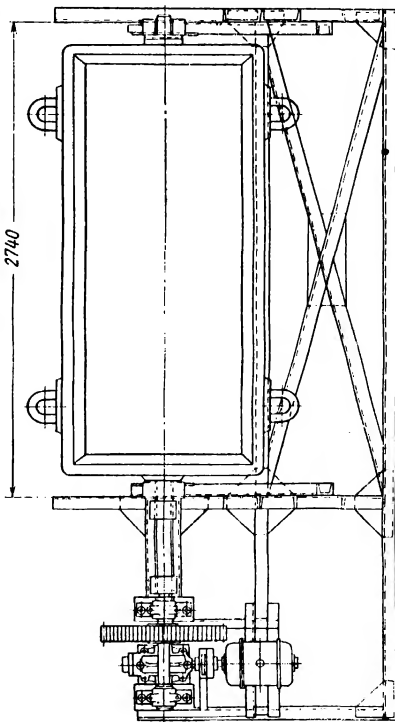
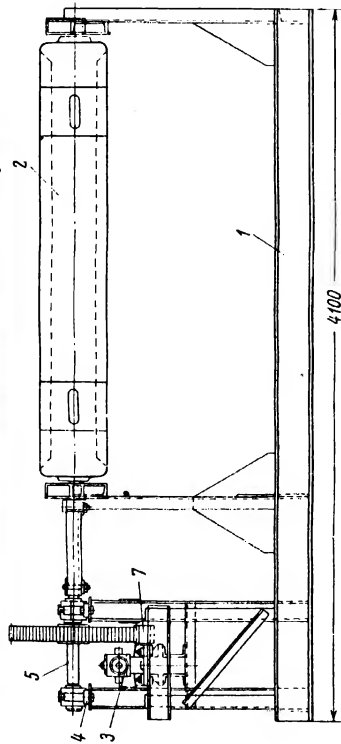


Fig. 68. Rotating mould for ferrosil-
icon teeming;

1—frame; 2—mould; 3—worm gear; 4—bear-
ing; 5—shaft; 6—gearing wheel; 7—pinion;
8—electric motor; 9—reducer coupling



Crystalline silicon, teemed into moulds, is dumped out into bins and conveyed to the cooling aisle, where it is cooled, crushed, cleaned and packed.

The composition of commercial ferrosilicon is given in Table 16.

Table 16

Alloy	Composition, %							
	Si	Fe	Al	Ca	P	S	Mn	Cr
Cr45	44.2	53.87	0.50	0.82	0.023	0.013	0.42	0.16
	45.6	52.78	0.67	0.41	0.025	Traces	0.38	0.14
	44.2	52.67	0.84	1.81	0.024	"	0.35	0.11
Cr75	46.2	51.81	0.74	0.70	0.020	0.01	0.12	0.40
	75.4	22.07	1.21	1.00	0.026	0.01	0.20	0.08
	76.5	21.05	0.72	1.45	0.027	Traces	0.19	0.06
	78.0	19.03	0.99	1.60	0.024	0.01	0.19	0.16
	77.5	20.01	1.04	1.10	0.028	0.01	0.20	0.11
Cr90	90.13	7.58	1.63	0.66	---	---	---	---
	88.67	8.60	1.90	0.83	---	---	---	---
	93.1	5.03	1.48	0.39	---	---	---	---
	91.69	5.98	1.79	0.54	---	---	---	---
Crystalline silicon	99.06	0.30	0.34	0.30	---	---	---	---
	98.98	0.27	0.33	0.42	---	---	---	---
	98.28	0.82	0.44	0.46	---	---	---	---
	99.01	0.40	0.29	0.30	---	---	---	---

Production Control

The quality of ferrosilicon is checked when the alloy is tapped out. Three samples are taken by putting a graphite crucible under the alloy jet: at the beginning, in the middle and at the end of tapping; the sample is sent to a shop laboratory for silicon assaying. The weight of the iron chips added is adjusted in conformity with the analysis results and a conclusion is drawn as to the correctness of technology.

A grading analysis is made with samples taken from cold ingots or with a mean sample taken during the pouring. A full chemical analysis is made of one ferrosilicon heat from each furnace once every ten days.

The correctness of technology during shifts is checked by hourly charging and specific power consumption time-tables. The amounts of charges smelted and power consumed are recorded on a special board every hour to ensure control. Deviations from a pre-set hourly time-table are indicative of abnormalities in the furnace run and call for the localisation of trouble and its elimination.

Process technology is also checked by electric metering instruments, by furnace-top symptoms and by the amount, physical

state and chemical composition of the slag tapped out with the alloy. It is recommended to weigh the slag each shift. The furnace chief and the shift foreman control the weighing of the charge periodically during the shift. Conformity of actual charging to that set by the time-table is checked daily by the chief foreman.

Correct reductant weight is calculated each shift by the foreman in accordance with actual coke moisture.

The quality of charge preparation is controlled continuously by the whole furnace crew.

The Quality Control Department exercises control over compliance with technological instructions, accuracy of charge weighing, the quality of the charge materials delivered to the furnaces, determines and eliminates the causes of low-grade raw material deliveries to the workshop, checks the quality of commercial products, determines and eliminates the causes of spoilage.

Ferrosilicon Smelting Production Data

Relatively cheap raw materials are used for the production of ferrosilicon; in this connection, the cost of electric power consumed comes to 30-45% of the over-all alloy cost. Because of this, the chief factor characterizing furnace operation is specific power consumption. Specific power consumption depends, in the first instance, on the grade of ferrosilicon smelted and increases with the silicon content. This is explained by the increased difficulty of reducing silicon when the iron content in the charge drops, as well as by increased silicon evaporation.

Silicon losses (in percentage of silicon charged into the furnace in the form of silica) amount to approximately 4 to 5% in the production of Cu45, 7 to 10% of Cu75, 15% of Cu90 and approximately 25% of crystalline silicon.

Prevention of silicon losses in ferrosilicon smelting is an important factor in the reduction of specific power consumption.

A decrease in silicon losses may be obtained by a proper choice of optimum electrical operating conditions, high-quality and stable composition of the charge and correct and attentive operation of the furnace-top and furnace taphole.

Lowering of alloy losses in tapping and dressing operations is also an important task.

Furnace down-time in ferrosilicon smelting causes a deterioration in the constant thermal and electrical operating conditions and sharply raises power consumption; hence, it is necessary to shorten furnace down-time and to effect repairs rapidly and efficiently.

All other conditions being the same, the larger the furnace the lower specific power consumption in ferrosilicon production.

The increase in furnace productivity as the result of a power rating rise, as well as other factors, is of considerable importance from the point of view of economy on wages and over-all shop and smelter expenses.

Consumption of raw materials, electrodes, anode paste, shells and electric power for the production of ferrosilicon is given in Table 17.

Table 17

Silicon Alloys Production Data

Material	Consumption per ton, in kg, for the grade of the alloy				
	Cu18	Cu45	Cu75	Cu90	Crystalline silicon
Quartzite	500	1,030	1,800	2,700	Quartz 2,900
Iron chips	900	620	210	215	—
Dry fine coke	225	500	860	975	—
Pitch coke	—	—	—	—	650
Charcoal	—	—	—	540	850
Anode paste	8.0	24	40	80	—
Graphitised electrodes	—	—	—	—	100
Steel plate	0.5	1.2	2.0	4.5	—
Steel rods	3.0	5.0	8.0	12.5	8.0
kWh					
Power consumption	1,900	4,500- 4,800	8,300- 9,000	12,300	14,000

Chapter 2

PRODUCTION OF SILICOCALCIUM

Designation and Composition of Silicocalcium

Complex reducers are widely employed in the production of high-quality steels; among these an important place is occupied by alloys of calcium with silicon generally known as silicocalcium.

Silicocalcium is used both in the form of powder (for the reduction of slag) and lumps.

Silicocalcium processing of steel promotes the latter's desulphurisation, as CaS which forms as the result of this process is a very stable compound and does not dissolve in steel.

The chemical composition of silicocalcium (ГОСТ 4762-49) is given in Table 18.

Table 18

Grade	Chemical composition, %				
	Calcium	Calcium + silicon	Aluminum	Sulphur	Phosphorus
	minimum		maximum		
Касп 0	31	90	1.5	0.04	0.05
Касп 1	28	90	2.5	0.04	0.05
Касп 2	23	85	3.0	0.04	0.05

Silicocalcium is delivered to consumers on a batch basis, packed in metallic drums.

Raw materials for the production of silicocalcium, their pre-smelting preparation. The mixture for the production of silicocalcium consists of quartzite, lime, fine coke, charcoal and coal.

Raw materials specifications are the same as in ferrosilicon-smelting practice.

Lime used for the production of silicocalcium should contain minimum 90 % of CaO. Poorly burned lime sharply raises power con-

sumption, lowers furnace efficiency, causes technology disturbances and shortens furnace life.

The presence of lime fines, which readily hydrate under the influence of atmospheric moisture as well as the moisture contained in reductants, is undesirable too. The size of lime lumps should range from 20 to 100 mm.

Physicochemical Principles of Silicocalcium Production

Calcium is an element of the second group of the Mendeleyev Periodic System (ordinal number 20) and is fifth as to its occurrence in nature; its content in the earth's crust is 3.25%. The greater portion of calcium occurs in nature as limestone, chalk, marble (CaCO_3), gypsum ($\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$), and dolomite ($\text{Ca} \cdot \text{Mg} \text{CO}_3$).

Industrial production of calcium is based upon the reduction of calcium oxide by aluminium in the presence of fluoric calcium at 1000-1160°C and in vacuum.

Calcium is a grey, lustrous metal with excellent plasticity. The chief physicochemical properties of calcium are the following:

Atomic weight	40.08
Specific gravity	1.54
Valence	2
Melting point, °C	849
Boiling point, °C	1487
Heat of fusion, cal/mol	2,230
Specific heat capacity at 25°C, cal/°C	0.157

Calcium is absolutely insoluble in iron.

Together with carbon it forms a stable carbide CaC_2 . The heat of formation of calcium carbide is 14,500 cal. The melting point is 2300°C.

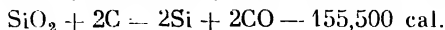
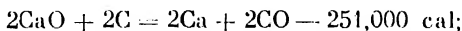
With silicon calcium forms silicides: Ca_2Si , CaSi and CaSi_2 . The most stable of calcium silicides is CaSi , which contains 58.8% Ca and 41.2% Si. The heat of formation of CaSi is 36,000 cal; its melting point is 1245°C. Silicide of calcium CaSi is a stabler compound than calcium carbide.

With oxygen calcium forms a stable oxide CaO . The specific gravity of calcium oxide is 3.4, its melting point is 2700°C. Formation of CaO from elements is accompanied by the evolution of a great amount of heat:



which is evidence of its stability.

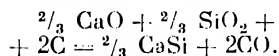
Silicocalcium is produced by the reduction of calcium and silicon from lime and quartzite by carbon reductants according to the following reactions:



Both reactions are given in relation to one molecule of oxygen for convenience of comparison. Considerable heat consumption is a requisite for the flow of the two above-mentioned reactions.

With the rise of temperature the stability of SiO_2 and CaO decreases, while that of CO increases (Fig. 69). Successful reduction of SiO_2 and CaO to silicon and calcium by carbon is possible only at temperatures when the resultant CO becomes stabler than the reduced oxides. This is attained, for SiO_2 , at temperatures above 1540°C and for CaO at temperatures above 2150°C .

When SiO_2 and CaO are reduced jointly the reduction process is facilitated by the formation of silicon and calcium which combine to form a stable compound—calcium silicide CaSi :



This reaction may take place at about 1600°C , i.e., at a temperature considerably below that required for the reduction of calcium oxide only.

In the production of silicocalcium silicon has the same function as iron in the production of ferresilicon: evaporation of calcium in the production process is prevented by the dissolution of calcium in silicon.

Calcium oxide and silicon, when available in the ratio of 35 to 55% CaO and 45 to 65% SiO_2 , form an easily fusible slag (melting point 1450 – 1550°C). The formation of such slag lowers the temperature in the reactive zone of the furnace and renders the reduction of calcium and silicon impossible; moreover, it attacks the protective layer and furnace lining. To prevent the formation of such slag the process is conducted with a great excess (30%) of reductant

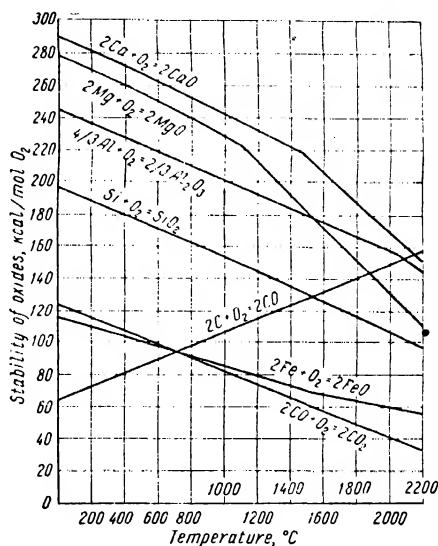
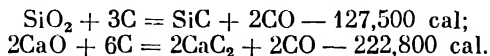


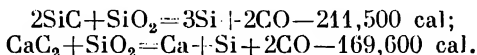
Fig. 69. Relationship between stability of oxides and temperature

over the theoretical amount necessary. This, in turn, brings about the formation of silicon and calcium carbides by the reactions:



These reactions consume less heat than the reactions of reduction to pure elements and, therefore, may proceed at temperatures below that at which slag is formed. Carbides, reverting to slag, raise its melting point and ensure the attainment of high temperatures in the reactive zone which are necessary for the successful reduction of silicon and calcium.

The excessive development of the carbide-forming process leads to the crusting of the furnace and, to avoid that, carbides are destroyed periodically by single additions of quartzite. The reactions of the process are as follows:



Slag assay in the production of silicocalcium varies widely. The average composition of normal slag is as follows: SiO_2 —54.36%; CaO —15.3%; SiC —8.8%; CaC_2 —15.0%; Al_2O_3 —2.5%; the rest— MgO , FeO and other components.

Charge Calculations

The main condition in the calculation of the charge for the smelting of ferrosilicon is prevention of the formation of excessive amounts of slag; the process is conducted with an excess of 30% of fixed carbon in the mixture over the amount theoretically necessary (the result is reliable duty of the lining due to the formation of a protective layer).

Assume that the following is the composition of the alloy: 30% Ca and 60% Si.

Assume further that the CaO content in the lime amounts to 90%, while that of SiO_2 in quartzite equals 97%.

Bearing in mind the considerable stack losses of calcium and silicon as well as smelting practice, let us take the following excess coefficients (in excess over those theoretically necessary) for the lime—2 times, quartzite—1.5 times, fixed carbon—1.3 times.

The carbon content in dry fine coke is 86%, and 63% in dry charcoal.

A charge portion comprises 65 kg of charcoal; the remaining portion of fixed carbon comes in the form of fine coke.

The amount of lime necessary for the production of 100 kg of alloy is:

$$\frac{30 \times 56 \times 2.0}{40 \times 0.90} = 93.5 \text{ kg.}$$

The amount of quartzite necessary for the production of 100 kg of alloy is:

$$\frac{60 \times 60 \times 1.5}{28 \times 0.97} = 199 \text{ kg.}$$

Assume that the following is the composition of a charge portion: 100 kg of lime and 200 kg of quartzite.

The necessary amount of fixed carbon per charge is

$$\left(\frac{100 \times 0.90 \times 12}{56} + \frac{200 \times 0.97 \times 24}{60} \right) \times 1.3 = 126 \text{ kg.}$$

The following amount of fixed carbon is introduced with charcoal:

$$65 \times 0.63 = 41 \text{ kg.}$$

The required amount of fine coke per charge:

$$\frac{126 - 41}{0.86} = 99 \text{ kg.}$$

Since the coal and fine coke delivered to the smelting shop have a considerable moisture content, the actual weight of the reductant per charge depends on the moisture content of the above-mentioned materials (Table 19).

Table 19

Reductant	Dry weight, kg	Moist weight, kg. with moisture in %									
		2	4	6	8	10	12	14	16	18	20
Charcoal	55	65	70	70	75	75	75	80	80	85	85
Fine coke	99	100	100	105	105	110	110	115	115	120	120

As a rule, the mixture is now composed as follows, in kg:

Quartzite	200
Lime	90-85
Fine coke	100-90
Charcoal	55
Anjer coal	30

Anjer coal was added to the burden after a series of tests and has played an important role in reducing alloy production costs, as it is approximately three times as cheap as charcoal.

Temporary variations in the composition of the charge given above may be tolerated in the following instances:

- a) when the furnace is switched over to silicocalcium smelting and when an excess of fixed carbon is required to create a carbide bedding on the hearth;
- b) when a sharp change occurs in the quality of lime the amount of lime in the charge may be altered;
- c) when the run of the furnace deteriorates.

Technology of Silicocalcium Production

Silicocalcium is smelted in furnaces of 10,000 kVA and higher rating, with secondary voltage equal to 115-123 V and electrode current equal to approximately 50,000 A. The choice of such electrical operating ranges is dictated by the desire to obtain the maximum possible deep and stable positioning of the electrodes, normally averaging about 700 mm.

In the case of silicocalcium smelting the furnace rapidly becomes crusted with carbides and has to be relined at least three times a year.

After current lining repairs the furnace is used for about 10 to 15 days, in smelting 45% ferrosilicon in order to heat it up. It is then cleaned by smelting lime. About 20 tons of lime are charged into the furnace to clean it completely; the duration of the melting is 10 to 12 hours. The melt obtained is tapped out into slag containers. If the bath is not cleaned satisfactorily an additional portion of approximately five tons of lime is introduced into the furnace; as soon as it is melted, a second tapping is effected.

After the bath-cleaning operation fine coke (3.5-4 tons) is charged on the hearth and the banks, following which the burden is charged slowly: the first eight hours at a rate of two charge portions an hour, the next eight hours at a rate of three charges an hour. During a shift the metal and slag are tapped two or three times. After a lapse of 16 hours normal charging comes into effect, at a rate of 1,570-1,650 kWh per charge.

The smelting of silicocalcium is a continuous process and a batch of mixture is charged as soon as the preceding portion slides down. A mixture with a considerable lime and charcoal content is porous and does not require pushing by poles. Instead of this it is imperative to rabble the centre of the charge column at the furnace-top with metal rabblers after each tap. This is done to prevent the

rise of the charge level, for it may cause rapid furnace crusting and shorten the latter's campaign. The most important task of the operator is to keep the charge level low for performance results depend on how this problem is solved.

Average-size charge cones are kept up around the electrodes to push off the evolving gases; this raises heat transfer efficiency and reduces the evaporation of calcium and silicon.

One symptom of abnormal calcium oxidation is the appearance of dark crimson flames at the furnace-top.

Normally there should be no gas blow-outs at the furnace-top and the flame should be uniform over the whole charge surface. The mixture should descend gradually, the quicker the nearer the charge section is to the electrode. When a large portion of the mixture collapses, the furnace operator pushes the mixture over the edges of the crater with a rabbler and then feeds the necessary amount of fresh burden.

The smelting of silicocalcium, as we have already pointed out, proceeds with a great excess of reductant. Consequently, the furnace hearth gets rapidly crusted with carbides of silicon and calcium and unless special measures are taken to destroy them, the furnace goes out of commission after a relatively short period of time and performance results drop sharply.

From 500 to 800 kg of quartzite are periodically charged into the furnace (usually three-four times per shift) to destroy carbides. Quartzite is charged after the tapping of metal, while the mixture is heaped beforehand around the electrodes to prevent quartzite from falling directly into the electrode crucible. Quartzite is charged around the electrode along its entire perimeter.

When the crusting of the furnace is rapid, quartzite is also charged by large portions (up to one ton).

The following are the symptoms of furnace crusting:

- a) high calcium content in the alloy (32% and higher);
- b) absence of slag during tappings or negligible slag fall; slag is viscous and high as to "volume test";
- c) tapping difficulties;
- d) high positioning of electrodes and characteristic buzzing of electric arcs;
- e) decrease in burden movement (furnace-top is bright) and frequent blow-outs.

In serious cases, apart from additional quartzite, there is a need for a decrease in reductant to improve the furnace run.

Lack of reductant is characterised by continuous current fluctuations (ammeter readings), speed-up in burden movement, increase in amount and improvement of running qualities of slag, lowering of the latter's calcium carbide content, decrease of the calci-

um content in the alloy, and by difficulties in taphole plugging.

When there is a lack of lime in the mixture the calcium content in the alloy drops and the running qualities of slag deteriorate. When there is an excess of lime, great amount of highly liquid slag is observed.

Whatever furnace trouble, it is necessary to check up on the preparation of the mixture as well as on the correctness of furnace operating procedures.

Tapping, Teeming and Dressing of Alloys

The alloy is tapped four times during a shift. Tapping is effected by means of a swing-aside receiver lined with firebrick. The receiver (Fig. 70), mounted on a carriage, has an upper cylinder equipped

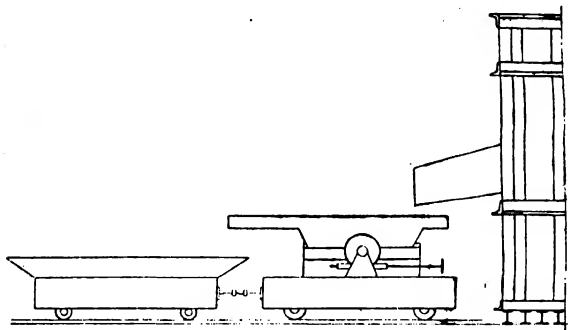


Fig. 70. Diagram of calcium-silicon teeming

with a pouring lip fettled with dry sand which helps to retain all tap slag in its recess. After the tapping the alloy is poured through the lip onto a cast-iron pan by tilting the receiver, while the slag, being heavier, remains in its recess.

Slag should not be allowed to penetrate into the alloy pan, for this contaminates the ingots and may spoil a large amount of alloy.

Taphole maintenance in silicocalcium smelting is similar to that in ferrosilicon smelting. The taphole should be poked at each tapping or, at least, lanced with oxygen; the appearance of flames through the taphole is evidence that this has been done properly. A non-standard alloy, clean from slag and crushed into lumps, may be added for resmelting into the pan before silicocalcium

is tapped. The amount of alloy to be resmelted should not exceed 100 kg per heat.

The composition of individual heats of commercial silicocalcium is given in Table 20.

Table 20

Composition of silicocalcium, %			
Si	Ca	Fe	Al
64.42	30.52	3.77	1.59
63.67	31.51	3.34	1.48
64.29	30.81	3.48	1.42
67.00	28.24	3.19	1.57

A metal ingot is extracted from the mould 1-1.5 hours after tapping, depending on its thickness. The slag is taken out of the receiver with the help of tongs after 30 to 40 minutes; large metal inclusions are sorted out from slag lumps after they have been cooled; smaller inclusions are picked out after the disintegration of the slag in the slag yard.

Cleaned and crushed alloy is packed by heat batches in metallic drums.

Production Control

The quality of silicocalcium is checked not only by the control methods generally used for silicon alloys and by its chemical composition, but also by the character of the fracture of the ingot. An alloy with less than 25% Ca has a uniform fine-grained structure. The fracture of the alloy becomes macrocrystalline when the calcium content exceeds 28-29%, while a columnar structure is observed when Ca in the alloy reaches 30% and over.

Besides this, the slag from each heat is tested for the calcium carbide content* (by the volume of acetylene generated, the so-called slag "volume" test).

Performance Results of Silicocalcium Production

One of the basic factors of silicocalcium production is specific power consumption. Average power consumption per ton of 28% Ca alloy comes up to 12,000 kWh.

Raw materials consumption per basic (28% Ca) ton of alloy is as follows, kg:

Quartzite (97% SiO_2)	1,900
Lime (90% CaO)	720
Charcoal	400
Anjer coal	240
Dry fine coke	670
Anode paste	230

The main ways of improving silicocalcium production performance results are the following:

- 1) choice of optimum electrical operating conditions;
- 2) improvement of raw materials quality;
- 3) prolonging the campaign of silicocalcium furnaces;
- 4) reduction of alloy losses which at present constitute a considerable value.

Chapter 3

MANGANESE ALLOYS

Use of Ferromanganese

Manganese alloys are those in which the predominant constituent is manganese.

The most important is ferromanganese—an alloy of iron and manganese, containing about 80% of Mn.

Ferromanganese is widely used in the manufacture of tool steels (up to 0.4% Mn) and structural steels (up to 0.6% Mn), and also of special steels with a high manganese content (up to 12-14% Mn), as well as an alloying addition.

Metallic manganese is used in the manufacture of special steels low in iron, low-carbon alloyed steels with a high manganese content, as well as in non-ferrous metallurgy.

Ferromanganese and silicomanganese are widely used reducers in the manufacture of steel. Ferromanganese is also employed for desulphurisation of steel, while silicomanganese is used as a semi-product in the manufacture of medium- and low-carbon ferromanganese and metallic manganese.

Grades of Manganese Alloys

The composition of ferromanganese as specified by State Standards is given in Table 21.

Ferromanganese of the grades Mn0, Mn1, Mn2, Mn3, and Mn4 is manufactured in electric furnaces, while that of grades Mn5 and Mn6 is manufactured in blast furnaces.

Stored carbon-bearing ferromanganese, containing over 82% Mn, disintegrates under the action of ambient moisture. Consequently, the manufacture of alloys with manganese above 81-82% is avoided.

Metallic manganese is used for smelting low-carbon special alloys with a low iron content. The latter's composition (as specified by GOST 6008-51) is given in Table 22.

Table 21

Chemical Composition of Ferromanganese and Its Use

ГОСТ	Denomination of alloy	Grade	Composition, %						Use
			Mn	C	Si	P		S	
			minimum	maximum					
4755-49	Electrothermic	Mn0 Mn1 Mn2	80.0	0.5	2.0	0.30	0.03	Deoxidiser and alloying addition for low-carbon steel	
	Low-carbon		80.0	1.0	2.0	0.30	0.03		
	Medium-carbon		80.0	1.5	2.5	0.30	0.03		
	Carbon	Mn3 Mn4	78.0	7.0	2.0	0.33	0.03	Ditto for steel with high-carbon content	
			76.0	7.0	2.0	0.38	0.03		
	5165-49	Blast-furnace	Mn5	75.1 and over	2.0	Group		0.03	Deoxidiser for high-carbon steel
A						B			
0.35						0.45			
Mn6			70.0-75.0	2.0	0.35	0.45	0.03		

Table 22

Chemical Composition of Metallic Manganese ГОСТ 6008-51

Grade	Mn. mini- mum	Chemical composition, %								Sum of impurities
		Si	P	Al	Fe	Cu	C	S		
		Maximum								
Mp 00	99.95	—	0.005	—	—	—	0.32	0.01	0.05	
Mp 0	99.70	—	0.01	—	—	—	0.10	0.10	0.30	
Mp 1	95.00	0.8	0.05	—	2.5	—	0.10	—	5.00	
Mp 2	93.00	1.8	0.07	—	3.0	—	0.20	—	7.00	
Mp 3	91.00	3.5	0.45	1.0	2.0	2.5	0.12	—	9.00	
Mp 4	88.00	4.0	0.50	1.5	3.0	4.0	0.15	—	12.00	

The composition of a complex silicomanganese reducer is given in Table 23.

Table 23

Chemical Composition of Silicomanganese (ГОСТ 4756-49)

Grade	Chemical composition, %			
	Si	Mn, minimum	C	P
			Maximum	
CHMH 20	20.0 and over	65.0	1.0	0.1
CHMH 17	17.0-19.9	65.0	1.7	0.1
CHMH 14	14.0-16.9	60.0	2.5	0.2

Physicochemical Properties of Manganese

Manganese Mn is an element of the seventh group of the Mendeleev Periodic System of Elements.

The earth's crust contains 0.090% Mn which is twelfth as to its occurrence among the elements.

Manganese is not found in elementary state in nature and occurs only in the form of compounds with other elements; these are mainly oxides, carbonates and silicates.

Pure manganese is a brittle silvery-white metal with the following physical properties:

Atomic weight	54.93
Specific gravity	7.2
Melting point, °C	1244
Boiling point, °C	2150
Valence	From 2 to 6

Manganese possesses a strong affinity to oxygen and sulphur.

Iron and manganese are soluble in all proportions in solid as well as in liquid state. Manganese does not give any chemical compounds with iron.

Manganese combines with carbon to give a stable carbide Mn_3C . Manganese carbide and liquid manganese are soluble in all proportions.

Together with silicon manganese forms several silicides: Mn_2Si , $MnSi$, Mn_2Si_3 ; the most stable among these is $MnSi$. Silicides of manganese are stabler compounds than manganese carbide, since the heat of formation of manganese silicide ($MnSi$) from elements is 14,000 cal while that for manganese carbide (Mn_3C) is 3,600 cal. Silicon, therefore, displaces carbon in its manganese compounds and gives birth to manganese silicide. The higher the silicon content in the alloy the lower the carbon content.

Fig. 71 shows the relationship between the amounts of silicon and carbon in silicomanganese. Manganese forms four compounds

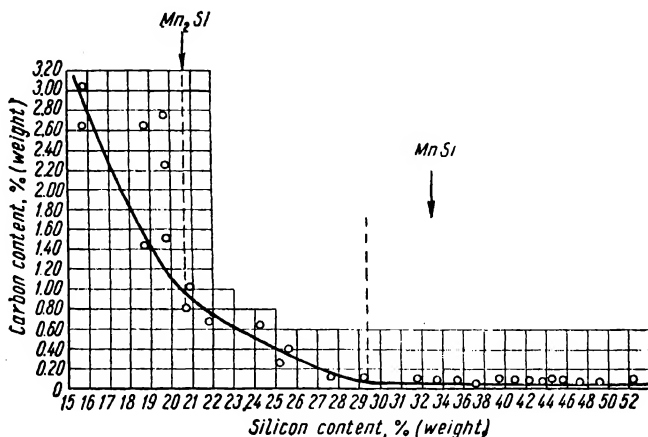
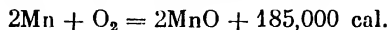
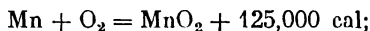


Fig. 71. Relationship between the carbon content in silicomanganese and the silicon content

with oxygen: manganese monoxide— MnO , manganese protoxide— Mn_3O_4 , manganese oxide— Mn_2O_3 and manganese dioxide— MnO_2 .

The formation of these oxides is accompanied by the generation of large amount of heat:



MnO_2 is a black amphoteric oxide; the remaining oxides are basic: Mn_2O_3 is blackish brown, Mn_3O_4 is dark-red.

The stablest oxide is manganese monoxide MnO which is deep green in colour.

Raw Materials for the Smelting of Ferromanganese

Manganese ores. The Soviet Union accounts for more than 50% of the world reserves of manganese ores. The largest deposits of these ores are in the Chiatura area in the Caucasus and the Nikopol area in the Ukraine.

It should be stressed that manganese ores are one of the chief

sources of raw materials used in metallurgy. Over 90% of world manganese ore output goes to the metallurgical industry.

Considerable deposits of manganese ores have been discovered in the last 15-17 years in the Urals, Siberia and Kazakhstan.

Among the industrial deposits of the Urals are the Polunochnoye, the Marsiaty and the Berezovsky group. In West Siberia the largest deposits are the Ussinskoye and the Mazulsky.

In Kazakhstan there is a group of Central Kazakhstan deposits which comprise the Atasuisky and the Djezdi deposits.

In the Far East surveyed deposits include iron-manganese ores at Khingan.

The ores in the eastern parts of the U.S.S.R. are considerably inferior to the Chiatura and Nikopol ores.

The main manganese minerals are pyrolusite MnO_2 —a mineral black in colour; manganite $\text{Mn}_2\text{O}_3 \cdot \text{H}_2\text{O}$ —grey steel in colour; psilomelane $(\text{MnO}, \text{BaO}, \text{CaO}, \text{MgO})\text{MnO}_2 \cdot n\text{H}_2\text{O}$ —light steel in colour; hausmanite Mn_3O_4 —brown-black; braunite Mn_2O_3 —brown-black, etc.

The chemical composition of ores and concentrates from the principal manganese deposits is given in Table 24. Considerable importance is attached to the assay of phosphorus in the ore, as phosphorus to a great extent reverts to the alloy and worsens its grade.

In ferromanganese manufacture the ores should be up to the following specifications: firstly, the manganese content in the ore should not be below 47% (productivity of the furnace and manganese recovery rise while specific power consumption drops with manganese assay increase in the ore); secondly, the Mn to Fe ratio should be higher than 8; thirdly, the silica (SiO_2) content in the ore should not be above 11%, otherwise manganese slag losses, specific power consumption and slag fall increase, while furnace efficiency drops.

Ores with a high silica (SiO_2) content may be used for the manufacture of silicomanganese.

Ordinary manganese ores are used for smelting metallic manganese in electric furnaces, while only very rich and pure uncontaminated manganese ores may be used for the aluminothermic method.

Fine metallurgical coke and coal are used as reducers in smelting carbon ferromanganese and silicomanganese.

Quartzites are used in the mixture to ensure a standard silicon content in silicomanganese.

To avoid slag increase the reducer should be low in ash content.

Silicomanganese is used as a reducer in the production of low-carbon grades of ferromanganese.

Iron chips are added to the mixture when carbon-bearing ferromanganese is smelted by a flux-free method.

Table 24

Chemical Composition of Manganese Ores and Concentrates from Chief Deposits

Deposit	Composition, %								
	Mn	P	SiO ₂	Fe	Al ₂ O ₃	CaO	BaO	MgO	H ₂ O
Chiatura, flushed	52-55	0.15-0.19	4-8	0.4-1.0	1.6-2.2	0.9-1.2	0.2-0.4	0.15-0.4	5-8
" ordinary	48-49	0.15-0.18	6-10	1.0-1.8	—	2-3	—	—	8-10
Nikopol, I grade	48-51	0.16-0.22	7.5-10.5	0.7-1.2	1.2-1.6	1.0-1.4	0.1-0.2	0.1-0.9	8-13
" II grade	42-44	0.17-0.25	13-15	1.5-1.9	2.0-2.5	2-3	—	1.5-2.1	12-16
Polunochnoye	21.7	0.22	26-27	3.3	2.92	5.51	—	1.24	—
" concentrate	31.0	0.19	—	2.5	—	—	—	—	—
Ussinskoye	19	0.14-0.23	19.6	7.6	—	1.23	—	3.26	—
" concentrate	21-24	0.17-0.23	12-16.0	5.0-8.0	—	12-14	—	—	—

Production of Carbon-Bearing Ferromanganese

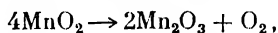
The production of carbon-bearing ferromanganese is a continuous slagging process.

Carbon-bearing ferromanganese is smelted in electric furnaces rated 7,500-13,500 kVA with transformer secondary voltage being approximately 120-135 V.

The oxides of iron, manganese, silicon, phosphorus, etc., contained in the ore are reduced by coke carbon in smelting carbon-bearing ferromanganese in electric furnaces.

The reduction of these oxides is attended by evolution of gases at the furnace-top.

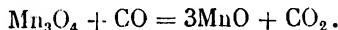
Higher oxides of manganese ores, introduced into the furnace, decompose with the evolution of oxygen at the low temperature of 430°C:



while manganese oxide decomposes at 950°C, according to the reaction

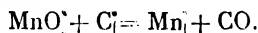


Mn_3O_4 is reduced by carbon monoxide to manganese protoxide by the reaction



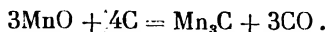
Consequently, manganese ore reaches the high-temperature zone as manganese protoxide.

The reduction of manganese protoxide by fixed carbon proceeds according to the reaction



This reaction takes place at about 1430°C.

At lower temperatures the reduction of manganese protoxide may be accompanied by the formation of manganese carbide, according to the reaction



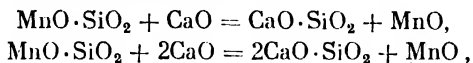
Manganese carbide obtained is reverted to the alloy. The latter's melting point is approximately 1250°C.

At these temperatures the conditions for the reduction of silica contained in the manganese ores are unfavourable and a considerable portion of it reverts to the slag along with the oxides of aluminium, calcium and magnesium.

Silicates $\text{MnO} \cdot \text{SiO}_2$ and $2\text{Mn} \cdot \text{SiO}_2$, generated in the furnace, revert to the slag and hamper the reduction of manganese protoxide.

A flux—lime—is added to the mixture to cut manganese slag

losses. In this case manganese protoxide is freed from the silicates according to the following reactions:



and may be reduced by carbon.

When smelting carbon-bearing ferromanganese, phosphorus pentoxide is readily reduced and reverts to the alloy, thus sharply worsening the latter's quality (about 65% of ore phosphorus is recovered in the alloy).

Carbon-bearing ferromanganese is manufactured by a flux-free method in order to obtain rich slags with a low-phosphorus content.

The advantages of the flux-free method are the following:

1) slags of the flux-free method of smelting are more fluid than those of fluxed heats;

2) amount of waste gases drops with the flux-free method;

3) production of carbon-bearing ferromanganese without the use of fluxes permits appreciably to increase furnace productivity and to lower specific power consumption from 3,600 kWh to 2,600-2,800 kWh;

4) subsequent use of rich slags permits to attain an over-all manganese recovery of 81-83% instead of 80% as is the case of the fluxing method;

5) slags, obtained through the flux-free smelting of carbon-bearing ferromanganese, carry a considerable amount of manganese (38-42%) and the negligible amount of phosphorus (0.015-0.030%), this permitting to use them as metallurgical raw materials instead of manganese ores in the smelting of low-carbon manganese alloys.

The slags possess a number of advantages over the manganese ore:

a) manganese occurs in the ore in the form of manganese dioxide, while in the slag it is in the form of protoxide; this makes it possible to use a lesser amount of reducers;

b) low phosphorus content in the slag (0.015-0.030% against 0.14-0.18% and more in the ore) ensures a sharp decrease in phosphorus in the alloys produced from slags.

When carbon-bearing ferromanganese is smelted by the flux-free method, the mixture is composed of manganese ore, metallurgical fine coke and iron chips.

Carbon-bearing ferromanganese is smelted with a secondary voltage of 120-135 V and a current density at the electrodes equal to 4-4.5 A/cm²; this ensures lower manganese evaporation losses from localised hot-spots.

Furnaces smelting carbon-bearing ferromanganese are lined like those smelting ferrosilicon.

The descent of the charge has a serious effect on the smelting

procedure; therefore, it is necessary to ensure a uniform burden movement over the whole furnace area; this is achieved by timely raking of the mixture onto the electrodes.

In carbon-bearing ferromanganese smelting a mixture batch consists of approximately 300 kg of manganese ore, 75 to 90 kg of fine coke and 15 to 20 kg of iron chips. For better furnace operation and adequate mixing of charge materials, coke is weighed first, manganese ore next and iron chips last.

As soon as the mixture sinks close to the electrodes, new portions of materials are fed to the furnace; the mixture is conveyed from furnace bins through chutes onto the furnace-top; after that the materials are heaped by rabblers (around the electrodes) in the form of cones rising 300-400 mm over the furnace-top level.

Normal furnace operation is characterised by the presence of constant charge cones around the electrodes; this ensures uniform evolution of gases over the entire charge surface, deep and stable positioning of electrodes in the mixture (1,200-1,500 mm), collapse-free burden descent, correspondence of the alloy and slag content to charge calculations, and easy tapping.

If the furnace works normally, the charge reaches the high-temperature zone well prepared and heated.

When the charge hangs on to the electrodes, there is a possibility of collapses with subsequent ejection of red-hot mixture and slag. High-temperature blow-outs (about 1000°C) which occur simultaneously may cause the breakdown of mechanical equipment as well as considerable manganese evaporation. Therefore, the sintered sections of the furnace-top should be poked by iron rabblers or poles, while the mixture hanging onto the electrodes should be pushed down.

Deviation from normal operating conditions may be caused by lack or excess of reducing agent in the mixture.

In the former case the alloy has low-silicon and high-phosphorus content, the electrodes are positioned excessively deep, power input at the electrodes fluctuates, manganese slag losses increase, furnace productivity falls and specific power consumption rises.

Lack of reducer may lead to the destruction of the furnace lining, especially in the taphole zone.

To set the furnace right raw-material proportioning (ore and fine coke) and the size of the reducer should be checked, additional amounts of reducer charged and, if necessary, coke weight per charge portion increased.

When a furnace operates with an excess of reducer, heat losses rise (for the electrodes are shallow) and evaporation of manganese increases. The silicon content in the alloy goes up. In this case, the size and accuracy of coke weighing should also be checked and manganese ore should, if necessary, be charged onto the electrodes.

In all cases of furnace trouble technological procedures at the furnace-top should be intensified.

In the production of carbon-bearing ferromanganese slag and metal are tapped out simultaneously through the same taphole six times per shift.

Metal is tapped into a dried ladle lined with firebrick. The alloy remains in the ladle while slag is lip-poured into cast-iron moulds arranged alongside it.

The taphole is cleared of slag following the tapping and is then packed as much as possible with fireclay. The ladle with the alloy is transported by a crane to the pouring side. If some slag remains in the ladle it is poured out; after that the alloy is teemed into whitened cast-iron moulds. An alloy sample is taken in the middle of the teeming operation to be analysed for manganese, silicon and phosphorus content.

After an hour or an hour and a half ferromanganese ingots are extracted from the moulds and sent to the finished-products floor where they are dressed, cleared of slag and shipped to customers.

Some smelters use the method of cascade teeming (Fig. 72) of carbon ferromanganese, but it complicates the subsequent dressing of the alloy due to slag contamination and subsequent alloy losses.

Raw-material, electrode and power consumption per basic ton of alloy (76% Mn) as well as the recovery of Mn from ore in the smelting of carbon FeMn by the flux-free method average:

Manganese ore (48% Mn), kg	2,800-2,620
Fine coke, sized, dry, kg	460-480
Iron chips, kg	95-120
Anode paste, kg	18-23
Electric power, kWh	2,600-2,800
Recovery of manganese, %	59-61

Furnace crews should concentrate on bringing down per ton losses of manganese and power consumption, as the raw materials and power are the highest cost items in the production of carbon ferromanganese.

It is possible to improve operating results in carbon ferromanganese smelting by using closed-top electric furnaces which bring down the cost price of the alloy and improve working conditions for furnace operators.

Smelting of Silicomanganese

Silicomanganese is widely used in metallurgy as a complex reducer and an alloying addition in the smelting of various grades of steel. Moreover, silicomanganese goes for the production of refined grades of ferromanganese.

The chemical composition of silicomanganese according to ГОСТ 4756-49 is listed in Table 23.

Silicides of manganese are stabler compounds than manganese carbides; therefore, the higher the silicon content in silicomanganese the lower is its carbon content.

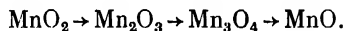
A 20% silicomanganese (Симн 20) is used for the manufacture of medium-carbon ferromanganese; a silicomanganese with approximately 30% of silicon is smelted for the manufacture of metallic manganese.

Silicomanganese can be produced by the following methods:

1) reduction of quartzite silica by coke carbon with the addition of manganese into the mixture (in the form of carbon ferromanganese);

2) simultaneous reduction of silica and manganese monoxide by the coke carbon contained in a mixture composed of quartzite and manganese-bearing slag.

The second method is more advantageous, as in this case silicomanganese is obtained by the simultaneous reduction of silica and manganese in the same apparatus, and that is more economical. Reduction from higher oxides to manganese monoxide proceeds along the same pattern as in the manufacture of carbon ferromanganese, i.e.,



Silicomanganese is usually manufactured by a continuous process in closed-top furnaces with deep penetration of the electrodes.

The upper layers of the mixture are heated by the escaping gases, evolving in the lower levels of the furnace. The furnace-top, therefore, is cold and losses through radiation are negligible; evaporation of manganese is also negligible. Silicomanganese with 20 to 25% silicon is easily obtained by this method.

Difficulties arise when the second method is used for the manufacture of alloys with a higher silicon content (over 25%). They are due to the fact that the silicon content in the slag goes up with the increase in the silicon content of the alloy, thus raising the viscosity of the slag and rendering its tapping difficult. It should be pointed out that these difficulties have been overcome by smelting the alloy in small furnaces with higher slag basicity and by introducing small amounts of fluorspar into the mixture.

The technology of smelting silicomanganese is described in the section "Production of Metallic Manganese".

Silicomanganese used for smelting refined ferromanganese is at present manufactured from a mixture composed of rich manganese-

bearing slag, quartzite, manganese ore, fine coke and iron chips. The introduction of manganese slag into the mixture permits the manufacture of silicomanganese with a minimum phosphorus content.

The charge has approximately the following composition, kg:

	Смш 14	Смш 17 and 20
Rich manganese-bearing slag (40% Mn, 30% SiO ₂)	—	300
Manganese ore (48% Mn)	300	100
Quartzite (96% SiO ₂)	80	80
Fine coke (dry)	93	90
Iron chips	16	20

Prior to being used rich manganese-bearing slag containing 38-44% Mn and 0.015-0.030% P is crushed into pieces maximum 50 mm in size.

Manganese ore may be used for the manufacture of silicomanganese with a high silicon content. The ore assays 48% Mn and 0.14-0.16% P.

Quartzites should contain minimum 96% SiO₂ and maximum 0.012% P; it is used in lumps not in excess of 60 mm; fines of less than 10 mm are screened out.

Silicomanganese of the Смш 14-20 grades is smelted in 7,000-10,000 kVA furnaces.

Burden charging and furnace-top operation are similar to those in the manufacture of carbon ferromanganese.

There is approximately 0.5-0.6 tons of slag per ton of alloy; slag composition is: 16-18% MnO, 42-45% SiO₂, 8-12% CaO, 11-12% Al₂O₃, 1.0-1.5% FeO and up to 1.5% MgO.

Alloy and slag are tapped out simultaneously, three to four times per shift.

Silicomanganese is tapped out into a dry ladle lined with fire-brick, and is then teemed into massive whitened moulds.

Each heat of silicomanganese is analysed for manganese, silicon, carbon and phosphorus content.

Following solidification (after a lapse of approximately 1-1½ hours) silicomanganese ingots are extracted from the moulds and conveyed to a finished-products' floor where they are crushed, cleared of slag, assembled into batches of similar analysis and shipped to consumers.

Raw-material, electrode and power consumption and the recovery of manganese in the production of silicomanganese are given in Table 25.

Table 25

**Raw-Material, Electrode and Power Consumption per Basic Ton
of Alloy and Recovery of Manganese in the Smelting of Silicomanganese**

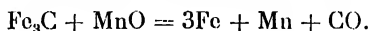
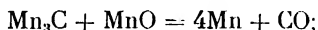
Materials consumed	СММН 14	СММН 17 and 20
Rich manganese-bearing slag (40% Mn), kg	—	1,440-1,500
Manganese ore (48% Mn), kg	1,950-2,050	690-720
Quartzite or chalcedony, kg	450-520	430-480
Fine coke, kg	650-720	450-490
Iron chips, kg	100-160	70-90
Anode paste, kg	24-27	25-29
Electric power, kWh	3,500-3,700	3,900-4,200
Recovery of manganese, %	69-71	69-70

Smelting of Refined Ferromanganese

Medium- and low-carbon ferromanganese is used for alloying manganese low-carbon steels.

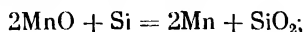
Medium-carbon ferromanganese may be manufactured by refining carbon ferromanganese by manganese ore in an electric furnace, by refining silicomanganese from silicon and by reducing manganese ore and specially smelted manganese-bearing slag.

The refining of carbon ferromanganese is described by the reactions



While the second reaction proceeds quite easily and completely at somewhat lower temperatures, the first is incomplete even at very high temperatures. The experiments in the decarburation of ferromanganese by manganese ore conducted by V. A. Bogolyubov have shown that ferromanganese with a carbon content of at least 2.5-3.5% may be obtained by this method at temperatures somewhat higher than 1700°C. This method is not practised at present because decarburation entails considerable manganese evaporation and the furnace lining is subject to severe operating conditions.

Widely applied in industry today is the method of manufacturing refined ferromanganese by refining silicomanganese from silicon by manganese ore according to the reactions



Lime is added to the mixture in the smelting of refined ferromanganese to bond the nascent silicon oxides to stable calcium silicates. Calcium oxide contained in the lime replaces manganese monoxide in its silica compounds and promotes its fuller reduction.

Introduction of excess lime leads to an increase in the amount of hard-to-smelt viscous slag as well as to greater evaporation of manganese. In practice, therefore, lime is added only to ensure the CaO to SiO_2 ratio within the limits of 1.1-1.3, with the manganese monoxide content in the slag being equal to 15-20%.

However, optimum slag basicity in the manufacture of medium-carbon ferromanganese by the above-mentioned method has not yet been sufficiently investigated.

Refined ferromanganese is smelted in three-phase rotating and rocking furnaces of 2,500 kVA at 111-178 V by the batch process with the complete melt-down of the mixture.

The furnace is lined with magnesite bricks. It is recommended to fuse the furnace hearth with magnesite, as the lining operating conditions are very severe due to the high-running qualities of metal and slag.

First-grade ore with 49-53% Mn and 0.14-0.16% P is used to refine silicomanganese; silicomanganese used in the manufacture of ferromanganese should contain 0.70-0.90% C (grade Mn1) and up to 1.4% C (grade Mn2). The silicon content in silicomanganese equals 18-22%, phosphorus should come to 0.1% and the size of the lumps should not exceed 30 mm. Lime should be hard-burned and in lumps not in excess of 80 mm; calcium and phosphorus should average at least 90% and maximum 0.1% respectively.

Silicomanganese richer in silicon (approx. 30%) and poorer in carbon (approx. 0.2-0.3%) should be used in the manufacture of low-carbon ferromanganese grade Mn0, containing less than 0.5% of carbon. As a rule, this alloy is replaced in steel smelting by medium-carbon ferromanganese and metallic manganese because it is difficult to manufacture it.

The composition of the mixture depends on the content of the raw materials used. The burden for a single heat consists approximately of the following components: manganese ore—4,600 kg, silicomanganese—3,000 kg, lime—2,300 kg.

As soon as the taphole is packed, power is applied to the furnace, the electrodes are slipped till they contact the remaining liquid slag, power input at the electrodes is adjusted and charging is started.

Power input should be increased gradually to prevent the metal remaining on the hearth from carburising.

The non-metallic part of the mixture, especially lime, is charged so as to protect the furnace lining from deterioration, i.e., onto

the walls. As the charge melts down, the materials situated at the walls are heaped round the electrodes."

A sample is taken following the complete melting-down of the charge with a view to determining the silicon content in the alloy. If the analysis shows that the alloy is of standard composition, it is tapped.

If the sample reveals a high silicon content, the alloy is held over in the furnace a little longer. When necessary, ore and lime are added. The metal is tapped into a ladle filled with slag from a preceding heat.

The excess of slag from the first ladle flows into the second ladle and then into the third. The second ladle, filled with slag, is left over for the tapping of the next heat.

During the tapping slag is sampled and analysed as to its SiO_2 , CaO , MgO , Al_2O_3 and FeO content.

Slag contains approximately 29-32% MnO , 30-32% SiO_2 , 29-36% CaO , 0.30-1% FeO , 1.2-2.0% Al_2O_3 , 0.02-0.03% P_2O_5 , and 1.0-0.70% MgO .

This slag is used as an addition to the burden of blast furnaces smelting pig iron.

The taphole is plugged with magnesite powder paste. Medium-carbon ferromanganese is teemed into whitened moulds. During the teeming the metal jet is sampled and analysed as to its manganese, carbon and phosphorus content.

When they have been cooled, ingots are conveyed to a finished-products floor where they are dressed and crated by heat batches for shipment to customers.

The over-all recovery of manganese from ore and rich manganese slag in the production of silicomanganese and medium-carbon ferromanganese amounts to 48-51%.

The main cost items in the production of medium-carbon ferromanganese are the raw materials and especially the silicomanganese. Furnace teams should consequently focus attention on bringing the manganese dressing and teeming losses down, as well as on raising silicon recovery from silicomanganese.

Consumption rates of materials per basic ton of medium-carbon ferromanganese, as well as the recovery of manganese are as follows:

Manganese ore (48% Mn), kg	1,340-1,420
Silicomanganese, kg	950-1,050
Lime, kg	760-850
Anode paste, kg	30-34
Power, kWh	1,600-1,900
Recovery of manganese, %	59-63

Smelting of Metallic Manganese

Metallic manganese is obtained by three methods: by the aluminothermic process, by the electrolysis of manganese sulphates and by the silicothermic (electrothermic) reduction of rich manganese slag.

The aluminothermic method is used to produce metallic manganese grades Mp3 and Mp4 (see Table 22); metallic manganese grades Mp00 and Mp0 are obtained by the electrolytic method; metallic manganese grades Mp1 and Mp2 are produced by the electrothermic method.

The aluminothermic method for the production of metals and alloys is based on the use of the chemical energy of reduction processes. As established by the Russian scientist S. F. Zhemchuzhny, the evolution of at least 550 cal of heat per gram of the mixture is necessary for the autogenous flow of the process. The mixture for the above-mentioned method of metallic manganese manufacture consists of manganese ore which, following roasting, should contain at least 57% Mn and no impurities. Secondary aluminium, powdered by means of the liquid state pulverisation process, is used as a reducer. Lime and fluorspar crushed into lumps less than 3 mm are added to the mixture to bond alumina and to lower viscosity of the slag. Smelting is done in a metal shaft lined with magnesite powder. A portion of the mixture is spread on the bottom, after which the priming charge, consisting of aluminium grains and unburned peroxide, is prepared and finally fired. The rest of the mixture is fed after the reaction sets in as the mixture melts down; the mixture should at all times be covered with a layer of slag. The heat takes 45 to 50 minutes. The tapping of the heat is withheld for 20 minutes for metal prills to settle in the slag; the slag is then tapped into a slag pot and the metal monolith is cooled and transported to a platform where it is dressed; as soon as its manganese, carbon, silicon and phosphorus content becomes known, it is packed in wooden or metal tare.

The metal contains 91-92% Mn, 2-3% Si, 0.2-0.5% Al, 0.38-0.40% P, 0.07-0.10% C, 1.7-2.0% Fe, 2.0-2.5% Ca.

Composition of the slag: 22.0-25.0% Mn, 2.5-3% SiO₂, 5-5.5% CaO, 70-72% Al₂O₃.

Consumption of materials in the case of the aluminothermic method per basic ton of metallic manganese (90% Mn) averages, kg:

Peroxide (55% Mn) . . .	2,300-2,400	Fluorspar	50-55
Secondary aluminium . .	560-600	Fuel oil	500-550
Lime	40-45		

Recovery of manganese in peroxide roasting and manganese smelting comes to 69-71%. The metal is of poor quality because of the

high phosphorus content (0.45%), and for that reason the production of aluminothermic manganese has been discontinued.

The electrolytic method of manufacturing metallic manganese is based on the electrolysis effect. Electrolysis is the process of the electric current conduction through the water solutions of salts, acids and alkalis, as well as through fused liquid salts, as the result of which certain elements and chemical compounds are deposited on the electrodes immersed in the solutions or melts.

In the electrolysis of manganese sulphate ($MnSO_4$) a thin layer of manganese is plated on the cathode. The metallic manganese stripped from the cathodes is melted in an electric induction furnace.

Electrolytic manganese is costly due to heavy consumption of expensive materials and power, and because of that its production is for the time being limited.

The use of the electrolytic method of manufacturing metallic manganese is advantageous when poor manganese ores are available.

At present the bulk of metallic manganese is produced by the electrothermic method. This method is complicated because the metal should contain very little phosphorus, carbon and iron, and because it calls for a multi-stage process consisting of three operations: a) production of high-manganese extra-low phosphorus slag; b) production of rich silicomanganese; c) reduction of slag oxides, produced in the first operation, by silicomanganese. This method permits the use of high-phosphorus ores.

Extra-low phosphorus slag rich in manganese is smelted by a full melt-down process in 2,000-2,500 kVA rocking three-phase electric furnaces with magnesite lining.

Phosphorus is reduced almost completely when manganese ore is smelted with fine coke.

Some quartzite fines are added to the mixture to raise slag fluidity and to promote separation of phosphorus ferromanganese prills.

Sometimes quartzite is partly replaced by silicomanganese slag. The composition of the mixture for the production of high-manganese extra-low phosphorus slag is as follows, kg:

Manganese ore (48% Mn)	8,100
Coke fines	950
Quartzite fines	650
Silicomanganese slag	850

After the melt-down of the mixture the heat is held in the furnace for 20-25 minutes before the slag is skimmed. The alloy obtained in this operation is tapped once or twice a day. Its composition is approximately as follows: 52-64% Mn, 28-37% Fe, 0.18-0.30% Si, 2.50-4.00% P, 4.5-6.0 C.

The chemical composition of the slag is approximately as follows, %:

Mn	SiO ₂	FeO	CaO	Al ₂ O ₃	MgO	P
67.2	24.92	0.20	3.19	3.14	1.24	0.018
62.5	26.70	0.59	2.56	4.96	2.20	0.012

Power consumption per ton of slag averages 1,300 kWh.

It has been established that silicomanganese used in the production of standard metallic manganese, should contain minimum phosphorus and iron and not more than 0.06-0.10% C.

The chemical composition of the silicomanganese produced at present is as follows (according to ferrous metallurgy specifications BTYЧМЗ 4-03-58): $\geq 28\%$ Si; $\leq 0.13\%$ C; $\leq 0.05\%$ P; $\leq 3.0\%$ Fe.

The following raw materials are used for the smelting of СММН 30/64:

- extra-low phosphorus slag;
- crushed quartzite;
- fine coke;
- lime;
- fluorspar.

Extra-low phosphorus manganese slag should contain at least 47% Mn and not more than 0.017% P. The lumps should not exceed 80 mm in size and there should be no prills of phosphorus-bearing metal in the slag. The size of quartzite lumps should be 10 to 80 mm while the SiO₂ content should not be below 96% and the phosphorus content should average 0.012%.

The size of coke lumps should be 10 to 25 mm, its ash content less than 12% and its moisture content not above 20%. Lime and fluorspar are used as fluidisers of acid slag. The size of lumps of lime is set at 10-80 mm, that of fluorspar below 50 mm.

The approximate composition of a charge portion for the production of the above-mentioned silicomanganese is as follows, kg:

Manganese extra-low phosphorus slag	150
Quartzite	57
Coke nuts, dry	53
Lime	17.8

Daily consumption of fluorspar is 120-150 kg. Melting is done by a continuous process in 2,500-kVA furnaces with line voltage on the low side being equal to 100-111 V.

The mixture is charged uniformly by small portions all over the furnace area with small heaps of mixture around the electrodes.

In normal conditions the gases should evolve uniformly all over the furnace-top. In the event of blow-outs the burden is poked with poles to give vent to gases.

Electrode positioning should be steady. The penetration depth of the electrodes should come to at least 500-600 mm from the furnace-top level.

Slag and alloy should flow out of the furnace uniformly. Increased slag viscosity and difficulties in tapping are signs of furnace trouble.

When the slag "boils" at the electrodes the time interval before the next tapping should be shortened and the slag tapped out as fully as possible.

To bring the reduction processes to an end, it is necessary to have high temperatures in the reaction zones; this is ensured by keeping the furnace-top closed and maintaining steady electrical operating conditions.

Slag and alloy are tapped out together three times per shift into a steel ladle or a ladle lined with firebrick.

It is not allowed to burn a taphole through with an iron rod. During tapping the eye of the taphole is poked by a slagged iron rod to ensure the complete evacuation of the slag.

Tapping takes 15 to 25 minutes.

After that the taphole is cleared of slag, packed with dry magnesite powder and sealed from the outside with refractory clay.

The alloy is held off in the ladle for 40 to 60 minutes to ensure the surfacing of the metal.

The alloy is teemed into well-whitened pans.

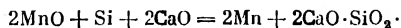
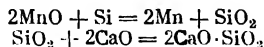
Solidified ingots are taken out of the pan and sent to store rooms where they are sampled for C, Si, Mn, Fe and P. The waste from ingot dressing is remelted.

Silicomanganese contains approximately 28% Si, 69% Mn, 2% Fe, 0.05% C, 0.04% P, 0.04% Ca, 0.01% Mg, 0.04% Al.

The slag obtained by the above-mentioned process has 39.6% SiO₂, 8.3% MnO, 24.0% Al₂O₃, 24.8% CaO, 3.1% Mg, 0.04% P, 0.16% FeO.

The last process in the smelting of metallic manganese is the refining of silicomanganese by an extra-low phosphorus manganese slag.

The process is as follows:



To increase furnace productivity and lower specific power consumption manganese slag is charged into the furnace in liquid state.

The manganese content in the middling slag should be minimum 47% and that of phosphorus maximum 0.017%. When solid slag is used its lumps should not exceed 60 mm in size.

Silicomanganese of СМН 30/64 grade is crushed down to 0-30 mm. It is recommended to use granulated slag.

The calcium oxide content in the lime should be above 90%, that of phosphorus below 0.008%.

Approximate charge calculations for the smelting of metallic manganese are given below.

The chemical composition of the metal is assumed to be: 97% Mn, 0.70% Si, 0.09% C, 1.8% Fe, 0.05% P.

The chemical composition of the raw materials is given in Table 26.

Table 26

Mixture components	Mn	SiO ₂	FeO	Al ₂ O ₃	CaO	MgO	P
Extra-low phosphorus slag (48% Mn) .	50	26	0.17	3.70	2.60	1.50	0.015
Lime	—	0.60	0.30	1.16	90.8	1.44	0.01
		Si	Fe	Al	Ca	Mg	
СЭМН 30/64	68	28	2.15	0.40	0.04	0.10	0.04

The distribution of the elements among the products of the process is shown in Table 27.

Table 27

Element	Reverted to (in % of over-all content in the mixture)		
	metal	slag	stack losses
Manganese:			
slag	42.5	47.5	10.0
silicomanganese	100.0	—	—
Iron:			
slag	90.0	10.0	—
silicomanganese	100.0	—	—
Mixture phosphorus	60.0	30.0	10.0

The silicon efficiency of silicomanganese is 70%, while manganese recovery from slags comes to 42.5%.

Let us assume that the electrode carbon has no part in the reduction of oxides.

The charge is calculated for 100 kg of slag.

The reduction of manganese monoxide to metal by the reaction



requires the following amount of oxygen:

$$\frac{0.425 \times 100 \times 0.5 \times 32}{110} = 6.2 \text{ kg.}$$

The following amount of silicon should be provided to bond oxygen (70% silicon efficiency):

$$6.2 \frac{28}{32 \times 0.7} = 7.75 \text{ kg.}$$

The necessary amount of silicon, including the 3.0% carried over to the metal, equals:

$$\frac{7.75}{0.97} = 8.0 \text{ kg}$$

or

$$\frac{8.0}{0.280} = 28.6 \text{ kg of silicomanganese.}$$

Let us calculate the weight of lime.

Amount of SiO_2 reverted to slag:

a) from oxidation of silicomanganese: -

$$7.75 \times \frac{60}{28} = 16.7 \text{ kg;}$$

b) from extra-low phosphorus slag (26% SiO_2):

$$100 \times 0.26 = 26.0 \text{ kg;}$$

amount of silica obtained:

$$16.7 + 26.0 = 42.7 \text{ kg.}$$

Amount of bases necessary with slag basicity being 1.6:

$$42.7 \times 1.6 = 68.3 \text{ kg.}$$

CaO carried over to slag from extra-low phosphorus slag:

$$100 \times 0.026 = 2.6 \text{ kg.}$$

To be introduced additionally: $68.3 - 2.6 = 65.7$ kg of CaO or $\frac{65.7}{0.908} = 72.4$ kg of lime.

Mixture composition, kg:

Extra-low phosphorus slag	100
Lime (90% CaO)	75
30/64 silicomanganese	28.6

The estimate of slag composition and weight is given in Table 28.

Table 28

Oxide	Reverts to slag, kg, from			Total	
	extra-low phosphorus slag	30/64 silicomanganese	lime	kg	%
SiO_2	$100 \times 0.26 = 26$	16.7	$75 \times 0.006 = 0.44$	43.14	28.50
CaO	$100 \times 0.026 = 2.6$	—	$75 \times 0.908 = 69.0$	71.6	46.83
MgO	$100 \times 0.015 = 1.5$	—	$75 \times 0.0144 = 1.08$	2.13	1.33

Oxide	Reverts to slag, kg, from			Total	
	extra-low phosphorus slag	30/64 silico-manganese	lime	kg	%
MnO	$100 \times 0.5 \times 0.475 \times \frac{71}{55} = 30.7$	—	—	30.7	20.072
Al ₂ O ₃	$100 \times 0.037 = 3.7$	—	$75 \times 0.0116 = 0.87$	4.57	3.00
P	$(100 \times 0.00015 + 28.6 \times 0.0004 + 75 \times 0.001) \times 0.3 = 0.01$	—	—	0.01	0.007
FeO	$100 \times 0.0017 = 0.17$	—	$75 \times 0.003 = 0.23$	0.40	0.261
				152.55	100.00

The estimate of metal composition and weight is given in Table 28a.

Table 28a

Element	Reverts to alloy, kg, from		Total	Metallic manganese assay, %
	extra-low phosphorus slag	silicomanganese		
Mn	$100 \times 0.5 \times 0.425 = 21.25$	$28.6 \times 0.68 = 19.5$	40.75	97.55
Fe	$\frac{100 \times 0.0017 \times 0.9 \times 56}{72} = 0.119$	$28.6 \times 0.0215 = 0.62$	0.739	1.77
Si	—	$28.6 \times 0.28 \times 0.003 = 0.24$	0.24	0.58
C	—	$28.6 \times 0.0009 = 0.025$	0.025	0.06
P	$(100 \times 0.00015 + 28.6 \times 0.0004 + 75 \times 0.001) \times 0.60 = 0.02$	—	0.02	0.04
			41.774	100.00

Amount of metal obtained (in basic weight):

$$41.774 \frac{97.55}{90} = 45.3 \text{ kg.}$$

Slag to metal ratio:

$$\frac{152.55}{41.774} = 3.66.$$

Specific consumption (per basic ton of metal):

$$\begin{aligned} \text{Extra-low phosphorus slag} & \dots \dots \dots \frac{100 \times 1,000}{45.3} = 2,208 \text{ kg;} \\ \text{Silicomanganese} & \dots \dots \dots 28.6 \times \frac{96.0}{94} \times 1,000 = 645 \text{ kg;} \\ \text{Lime (90\%)} & \dots \dots \dots \frac{75 \times 1,000}{45.3} = 1,630 \text{ kg.} \end{aligned}$$

Recovery of manganese from the mixture:

$$\frac{900 \times 100}{2,208 \times 0.48 + 645 \times 0.64} = 61.0\%.$$

Metallic manganese is smelted in batches in rocking rotating furnaces. The smelting comprises the following stages:

- a) lime and slag charging;
- b) power input build-up;
- c) melt-down of the lime-slag mixture;
- d) reduction period which includes reduction of manganese monoxide (contained in the slags smelted) by silicomanganese silicon added to the melt.

The slag to be smelted weighing from 6 to 7 tons is charged in liquid state; when the slag is solid it should weigh approximately 3.5 tons.

When working with liquid slag it is first necessary to charge 25% of the weight of the lime, heaping it onto the walls and spreading it on the hearth (to protect the lining); following this a first portion of liquid slag is charged (from one ladle) and power input is gradually increased. When the power input has reached its normal level and the second portion of slag has been poured in, the remainder of lime is charged in accordance with the weight of slag. This method considerably speeds up the smelting procedures, decreases manganese evaporation losses and improves the silicon efficiency as a reducer.

At the beginning the smelting is effected with secondary voltage of 356 V. After the appearance of long electric arcs the furnace is changed over to 222 V. Silicomanganese is charged in several batches in the course of the heat; after the last portion has been fed the heat is held off for 15 to 20 minutes with the power on and the metal and slag are then tapped out.

Tapping is effected into an unlined steel ladle filled with slag from the preceding heat.

Two or three tappings are made per shift.

After the slag has been poured out the ladle with the metal is vacuummed in a special installation to obtain a dense ingot with no gas cavities, as well as to cut down the amount of dressing scrap. Following the vacuum processing the metal is held off in the ladle to bring its temperature down and to solidify the slag completely; it is then teemed into whitened metallic moulds.

The metallic manganese produced by this method contains 97.1% Mn, 0.64% Si, 1.39% Fe, 0.052% P, 0.08% C, 0.34% Al, 0.10% Ca, 0.36% Mg.

The chemical composition of the slags obtained in the smelting of metallic manganese is as follows: 28.44% SiO₂, 43.3% CaO, 22.19% MnO, 2.13% Al₂O₃, 2.27% MgO, 0.46% FeO, 0.02% P₂O₅.

Over-all recovery of manganese from extra-low phosphorus slag and silicomanganese averages 63-64%.

Chapter 4

PRODUCTION OF FERROCHROME

Use of Ferrochrome

Ferrochrome is an alloy of chrome and iron with additions of carbon, silicon and several other elements.

As an alloying element, chrome is a component of numerous grades of steels and special alloys. Steels alloyed with chrome acquire valuable physical properties. It increases steel hardness, strength, yield point and elasticity.

Thanks to its positive effect on mechanical, physical and chemical properties of steel chrome is widely used in the manufacture of structural, tool and special steels (acid- and heat-resisting, stainless).

Low-carbon steel with more than 12% of chrome and approximately 8% of nickel is called stainless steel.

When steel contains chrome and silicon it acquires heat-resisting qualities.

Chrome is used most as an alloying element in combination with nickel (17.5-19.0% Cr and 8-11% Ni). Such a combination of elements gives steel acid-resisting characteristics. Alloys of the nichrome type possess high electric and anti-oxidation resistance properties and are used as electric heating elements (60-80% Ni and 15-20% Cr). Heat-resisting steels containing 13-60% Cr are widely used in the manufacture of gas-turbine and jet-engine parts.

Chromous steels containing approximately 1% of carbon and 1% of chrome are very hard and are widely used in the manufacture of ball and roller bearings.

Grades of Ferrochrome

Ferrochrome is manufactured (Table 29) in various grades for the alloying of alloys and steels.

Metallic chrome with specifications in accordance with GOST 5905-51 is used for the manufacture of steel with special physical properties and alloys with a low iron content (Table 30).

Table 29

Composition of Ferrochrome

Class of ferrochrome	Grade	Cr, minimum	C	Si			P	S
				low-silicon	medium-silicon	silicon		
				maximum				
Extra-low carbon	Xp0000	65	0.05	1.0	1.5	—	0.06	0.04
	Xp000	65	0.07-0.10					
	Xp00	60	0.11-0.15					
Low-carbon	Xp0		0.16-0.25					
	Xp01	60	0.26-0.50	1.5	2.0	3.0	0.06	0.04
Medium-carbon	Xp1	60	0.51-1.0					
	Xp2		1.1-2.0	—	2.5	3.0	0.10	0.04
	Xp3		2.1-4.0					
Carbon	Xp4	65	4.1-6.5	2.0	3.0	5.0	0.07	0.04
	Xp6		6.6-8.0					

Table 30

Composition of Metallic Chrome

Grade	Chemical composition, %							
	Cr, minimum	Fe	Al	Si	Cu	C	P	S
		maximum						
X0	98.5	0.6	0.5	0.4	0.06	0.02	0.02	0.02
X1	98.0	0.8	0.7	0.5	0.06	0.05	0.03	0.04
X2	97.0	1.2	0.8	0.5	0.1	0.06	0.05	0.05

In some cases special grades of extra-low carbon and nitrided ferrochrome are used; the composition of these grades of ferrochrome is specified by GOST 4757-49, given in Table 31.

Table 31

**Composition of Extra-Low Carbon and Nitrided Ferrochrome
and Chrome Aluminium Alloy**

Class of ferrochrome	Grade	Chemical composition, %						
		Cr, mini- mum	C	Si	Al	P	S	N, minimum
Extra-low carbon	Xp01	70.0	0.04	0.08	0.6	0.02	0.3	—
	Xp02	70.0	0.04	1.0	0.7	0.03	0.3	—
Nitrided	XpH 1	70.0	0.05	1.0	0.7	0.03	0.03	0.90
Chrome alumin- ium alloy	XAlI	60.0	0.04	1.0	minimum 16.0	0.02	0.03	—

Here is how the various grades of ferrochrome are used in the manufacture of alloyed steels:

Xp0000—for the manufacture of high-chrome heat-resisting steels and soft stainless steels with carbon below 0.07%;

Xp000—for the manufacture of high-chrome heat-resisting steels with up to 0.20% carbon, as well as for stainless and acid-resisting steels;

Xp00—for the manufacture of acid-resisting stainless steels with carbon up to 0.3% and several grades of low-carbon chromium-tungsten steel;

Xp0—for the manufacture of chromous stainless steels;

Xp01 and Xp1—for the manufacture of low-carbon structural steels;

Xp2 and Xp3—for the manufacture of structural and chromous tool steels with carbon content up to 0.50%;

Xp4 and Xp6—for the manufacture of all grades of chromous steels with a high carbon content (tool steels).

Physicochemical Properties of Chrome

Chrome (Cr) is an element of the sixth group of the D. I. Mendeleev Periodic System of Elements.

Mean chrome content in the earth's crust comes to 0.02%.

As an element, chrome was discovered in 1797 simultaneously by Vauquelin and Klaproth.

Pure chrome was obtained for the first time in 1854 by the electrolysis of aqueous solutions of chrome chloride. In 1908 a metal containing 99.9% Cr was obtained by the reduction of chrome oxide by hydrogen.

Ferrochrome was obtained for the first time in 1820, while carbon-bearing ferrochrome with 60% Cr and 6% C was produced in an electric furnace in 1893. Low-carbon ferrochrome and metallic chrome were initially obtained by the aluminothermic method.

Pure chrome is a grey lustrous metal.

Chrome possesses the following physicochemical properties:

Atomic weight	52.01
Specific gravity at 20°C, g/cm ³	7.19
Valence	2; 3; 6
Melting point, °C	1930
Boiling point, °C	2470
Specific heat capacity at 25°C, cal/°C	0.12

Chrome and iron are soluble in all proportions in liquid as well as in solid states.

Together with carbon chrome forms stable carbides: Cr₄C with a melting point of 1520°C and 5.46% C; Cr₇C₃ with a melting point of 1780°C and 9.01% C; Cr₂C₃ with a melting point of 1895°C and 13.34% C.

Carbon ferrochrome contains a double carbide (Cr, Fe)₇C₃. Low-carbon and extra-low carbon grades of ferrochrome also contain carbon in the form of double carbide (Cr, Fe)₄C. There also is a double carbide (Cr, Fe)₃C in the Cr-Fe-C system.

With silicon chrome forms silicides of chrome: Cr₃Si with a melting point of 1710°C, Cr₃Si₂, CrSi₂ with a melting point of 1550°C and the most stable silicide CrSi with a melting point of 1545°C.

Chrome gives three oxides with oxygen. Basic oxide CrO (chrome monoxide) is unstable in the air and turns rapidly to Cr₂O₃.

Amphoteric oxide Cr₂O₃, or chrome oxide, is lustrous-green in colour; its specific gravity is 5.21 g/cm³ and its melting point is 2275°C.

Acid oxide CrO₃ (chromic anhydride) is crimson in colour, has a specific gravity of 2.7 g/cm³ and a melting point of 196°C. Chromic anhydride decomposes entirely at 240°C with the formation of chrome oxide Cr₂O₃.

Raw Materials for the Smelting of Ferrochrome

The following materials are used to smelt various grades of ferrochrome in electric furnaces: chrome ores, reducers (fine coke and ferrochrome silicon), fluxes (quartzite and lime).

Chrome ores, whose main chrome-bearing components are the chrome-spinels (Mg, Fe) (Cr, Al, Fe)₂O₄, containing from 48 to 62% of Cr₂O₃ and the rest being gangue formed by Al₂O₃, FeO, MgO, etc., are used industrially. The presence of gangue affects the composition of chrome ores: it changes the chrome oxide content in the miner-

al as well as the chrome oxide to iron monoxide ratio—the main factors of chrome ore quality.

Since iron monoxide is always more readily reduced than chrome oxide, iron is wholly reverted to the alloy when chrome is reduced.

Although chrome ore deposits are widespread, only a limited number are of industrial interest.

The Soviet Union has extremely rich reserves of chrome ores, particularly in Kazakhstan and the Urals, where their mining is concentrated.

Ores from the Ural deposits (the most important is the Saranovsky deposit) have a somewhat lower chrome oxide content (20-40%), as well as a negligible Cr_2O_3 to FeO ratio, and because of that they are not used at present for the smelting of ferroalloys.

The ferroalloy industry uses chrome ores of the Donskoye deposit which was discovered near Aktyubinsk (the Kazakh S.S.R.) in 1936.

The chrome ores of the Donskoye deposit consist of chrome spinels (of constant composition) and its gangue is composed mainly of serpentine.

The approximate composition of Donskoye ore chrome spinels is as follows: 61% Cr_2O_3 , 15% FeO , 14.2% MgO , 9.8% Al_2O_3 .

The brownish colour is imparted to the ore by iron ochres. The chrome ores of the Donskoye deposit vary widely as to their chemical composition and physical state. The approximate composition of the ores is as follows: 44-62% Cr_2O_3 , 12-18% iron oxides (evaluated in FeO), 0.5-10% SiO_2 , 11-20% MgO , 6-13% Al_2O_3 .

The ores of the Donskoye deposit generally possess a high Cr_2O_3 to FeO ratio, exceeding 3.0; this permits smelting ferrochrome containing more than 70% Cr.

By their physical state the Donskoye ores are subdivided into lumpy, loose fragmental, friable and powder-like ores. Ochre ore with a high iron content permits obtaining foundry ferrochrome with more than 2.5-3.0% of silicon and 61-62% of chrome. Ores with 10% SiO_2 cannot be used for the manufacture of foundry ferrochrome because of their low melting point, which leads to the disintegration of the burden and to furnace trouble. For this reason the ore, used for the smelting of grade Xp4 and Xp6 ferrochrome should not contain more than 5% of silica.

Ores used for the production of refined ferrochrome by the silico-thermic method should have the minimum possible SiO_2 content for otherwise additional quantities of fluxes have to be provided and that entails additional losses of chrome and electric energy. The iron oxide content should be at the minimum too, since a high FeO content leads to unwarranted consumption of high-cost reducer (ferrochrome silicon) and damages furnace lining.

According to the ferrous metallurgy chrome ore specifications

ЧМТУ 5775-57, ores intended for the manufacture of ferroalloys must meet the requirements (as to chemical composition and physical state) listed in Table 32.

Table 32

Data	Grades		
	ДХ-1	ДХ-2	ДХ-3
Content, %:			
Cr ₂ O ₃ , minimum	52	50	45
SiO ₂ , maximum	5	8	10
P mean, per shipping batch	0.02	0.02	0.02
P in a batch, maximum	0.03	0.03	0.03
Cr ₂ O ₃ / FeO ratio, minimum	3.2	3.0	2.7
Size of lumps, mm	10 300	0-300	0-300
Fines 0-10 mm maximum, %	15	—	—

There should be no carbon-bearing materials in the chrome ore. The permissible weight of gangue is 1.2% of the total.

Chrome ore is delivered in batches which should be composed of similar grades of ore. The weight of one batch should not exceed 240 tons.

Fine coke, whose characteristics were given in the description of ferrosilicon-smelting process, is used as a reducer in the production of carbon and foundry ferrochrome.

Ferrochrome silicon described below is used as a reducer when smelting other grades of ferrochrome.

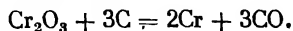
The fluxing material used in the manufacture of refined ferrochrome is lime. It should be well burned and contain at least 90% CaO and below 0.025% P. The lumps of lime should not exceed 80 mm in size. Hydrated lime is not to be used.

Quartzites are used as a fluxing material when smelting carbon grades of ferrochrome. The composition and characteristics of quartzites were described in the chapter "Production of Silicon Alloys".

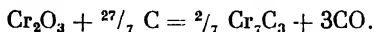
Production of Foundry and Carbon Ferrochrome

Chrome ore is used for the manufacture of carbon ferrochrome; oxides contained in the ore are reduced by carbon reducers (fine coke).

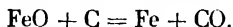
Chrome oxide is reduced by carbon at 1230°C with heat absorption according to the reaction



Stable carbides are formed in the reduction of chrome oxide by carbon at 1100°C according to the reaction



The reduction of iron monoxide proceeds according to the reaction



Iron reduced from ore promotes the reduction of chrome, since chrome carbides, which appear in the latter process, dissolve in iron with the formation of double carbides $(\text{Cr}, \text{Fe})_7\text{C}_3$.

The melting point of commercial carbon ferrochrome is about 1500°C.

Although the temperatures of the reduction processes of chrome and iron oxides are relatively low, the smelting of carbon ferrochrome is practically effected at 1650-1750°C; this ensures sufficient slag and alloy fluidity and satisfactory slag-alloy separation.

The choice of slag composition is an important part in carbon ferrochrome-smelting technology. The composition and melting point of the slag depend almost entirely on the composition of chrome ore gangue. The main slag components in carbon ferrochrome smelting are silica SiO_2 , magnesium oxide MgO and alumina Al_2O_3 .

The composition of slags in the smelting of carbon grades of ferrochrome is approximately as follows, %:

	SiO_2	MgO	Al_2O_3	CaO	FeO	Cr_2O_3
Xp4	25-32	32-38	27-30	1-4	1-2	3-6
Xp6	27-33	28-35	25-30	1-4	1-2	3-6

The chemical composition of the alloy produced is as follows, %:

	C	Cr	Si
Xp4	5.3	74	0.7
Xp6	7.2	69	1.8

Partial refining of alloy from carbon by chrome oxide according to the reaction: $\text{Cr}_7\text{C}_3 + \text{Cr}_2\text{O}_3 = 9\text{Cr} + 3\text{CO}$ is effected in the process of smelting of grade Xp4 carbon ferrochrome.

This process is possible only at high temperatures and with elevated concentrations of chrome oxide in the reaction zone. Consequently, the smelting process is effected in the presence of hard-to-fuse slags which are formed when hard-to-reduce lumps of rich ores are used.

Alloys are partially refined due to the formation of the so-called ore layer at the slag-alloy interface. This layer is met with when certain grades of ores with hard-to-fuse gangue are used. When the alloy and slag are tapped, the layer remains in the furnace because it is quite viscous. Drops of alloy trickle down through the layer and are partly refined from carbon.

Unlike carbon ferrochrome, foundry ferrochrome has a low chrome content (60-65%) and an elevated silicon content (average 2-3%). Its carbon content is not specified and this may cause an excess of silica and reducer in the mixture. Such an alloy assay permits to use low-quality ores (grades JX-3), chrome-bearing slags (25-30% Cr_2O_3) and tailings of ferrochrome and ferrosilicon smelting.

The high temperatures of the process ensure the reduction of silicon in addition to that of iron and chrome. The amount of silicon reduced depends on the presence of silica in the slag and reducer excess in the mixture.

Slags obtained in the smelting of foundry ferrochrome contain approximately 31-37% SiO_2 , 23-31% MgO , 29-36% Al_2O_3 , 2-3% CaO , 3-5% Cr_2O_3 , 1-2% FeO .

The chemical composition of foundry ferrochrome is approximately as follows: 7.5-8.5% C, 59-64% Cr, 0.5-4.0% Si.

Quartzites are added to the mixture to provide the necessary amount of silica in smelting both carbon and foundry ferrochrome.

A simplified method of charge calculations for the smelting of grade Xp6 carbon ferrochrome is given below.

Charge Calculations

Initial data:

Reduction of chrome oxide in the ore, %	90
Reduction of iron oxides in the ore, %	95
Carbon content in the alloy, %	7
Silicon content in the alloy, %	1.5

The amount of fixed carbon in the mixture exceeds the stoichiometric amount by 25%.

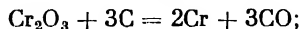
The SiO_2 content in the slag (assuming that the slag is composed of three components only) comes to 30%.

The composition of the ore (Aktyubinsk deposit) is as follows: 54.0% Cr_2O_3 , 15.6% FeO , 13.6% MgO , 10.0% Al_2O_3 , 4.5% SiO_2 .

The reducer—fine coke—contains 87% of fixed carbon. Quartzite, containing 97% SiO_2 , serves as a flux.

The charge calculation is made for one portion of the mixture (500 kg of moist or 475 kg of dry Aktyubinsk ore):

amount of chrome reduced by the reaction



$$\frac{475 \times 0.54 \times 0.90 \times 104}{152} = 158 \text{ kg};$$

amount of iron reduced by the reaction



$$\frac{475 \times 0.156 \times 0.95 \times 56}{72} = 54.8 \text{ kg.}$$

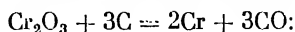
Chrome and iron total: $158 + 54.8 = 212.8 \text{ kg.}$

The chrome and iron content in the alloy (the sum of carbon and silicon is subtracted) averages: $100 - (7 + 1.5) = 91.5\%$.

The amount of alloy obtained: $\frac{212.8}{0.915} = 232.6 \text{ kg;}$ the amount of silicon in it: $232.6 \times 0.015 = 3.5 \text{ kg.}$

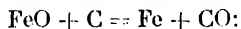
Amount of carbon necessary:

for the reduction of chrome oxide according to the reaction



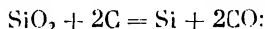
$$\frac{158 \times 36}{104} = 54.7 \text{ kg;}$$

for the reduction of iron monoxide according to the reaction



$$\frac{54.8 \times 12}{56} = 11.7 \text{ kg;}$$

for the reduction of silicon oxide according to the reaction



$$\frac{3.5 \times 24}{28} = 3.0 \text{ kg;}$$

for the carbonisation of the alloy:

$$232.6 \times 0.07 = 15.3 \text{ kg.}$$

Total amount of carbon necessary:

$$54.7 + 11.7 + 3.0 + 15.3 = 84.7 \text{ kg.}$$

Amount of fine coke necessary (including the 25% excess):

$$\frac{84.7 \times 1.25}{0.87} = 121.7 \text{ kg.}$$

Let us calculate the necessary amount of quartzite (x being the amount of quartzite).

Introduced with the ore:

$$\begin{array}{ll} \text{Al}_2\text{O}_3 & \dots \dots \dots 475 \times 0.10 = 47.5 \text{ kg} \\ \text{MgO} & \dots \dots \dots 475 \times 0.136 = 64.6 \text{ kg} \\ \text{SiO}_2 & \dots \dots \dots 475 \times 0.045 + 0.97x - \frac{3.5 \times 60}{28} = 13.9 + 0.97x. \end{array}$$

$$\begin{aligned} \text{Al}_2\text{O}_3 + \text{MgO} + \text{SiO}_2 &= 47.5 + 64.6 + (13.9 + 0.97x) = \\ &= 126.0 + 0.97x, \end{aligned}$$

$$13.9 + 0.97x = 0.3(126.0 + 0.97x),$$

$$13.9 + 0.97x = 0.3 \times 126.0 + 0.3 \times 0.97x,$$

$$x = \frac{0.3 \times 126.0 - 13.9}{0.97 - 0.3 \times 0.97} = \frac{23.9}{0.68} = 35 \text{ kg.}$$

Thus, the composition of the mixture for the smelting of Xp6 grade carbon ferrochrome will be as follows, kg:

Chrome ore	475
Fine coke	121.7
Quartzite	35

Carbon and foundry ferrochrome are smelted in three-phase electric furnaces of 7,500-10,500 kVA at a line voltage of 130-154 V. The smelting of carbon ferrochrome is a continuous process with a closed top.

The mixture is charged into the furnace, mainly onto the electrodes, as the preceding portion melts down.

The charge mixture is composed of 500 kg of chrome ore, stoichiometric amounts of quartzite and fine coke, and 30-40 kg of reverts. In the smelting of ferrochrome there may occur sulphur waste and because of that the process is conducted with a hot furnace-top to ensure the evaporation of sulphur; high-sulphur fine coke is not to be used. Power consumption per one batch of charge should average 925 kWh in normal conditions.

Uniform charging, stable positioning of electrodes, steady uniform light-yellow flame at the top and easy tapping of alloy and slag and their correct composition are proof of normal operating conditions.

The following furnace troubles may develop in smelting carbon ferrochrome:

1. Lack of reducer in the mixture (the amount of quartzite being normal) causes fluctuations of input at the electrodes and high consumption of the electrodes. The carbon content in the alloy decreases, slags become viscous and their chrome oxide content goes up. A single charge of fine coke (200-300 kg) and an increased amount of reducer in the mixture remedy the situation.

2. Excess of reducing agent, with correct proportioning of quartzite, leads to high positioning of the electrodes and evolution of white flames at the top. The silicon content in the alloy increases while the silica, chrome oxide and iron monoxide content in the slag falls.

The slag is overheated.

To set the furnace right the amount of fine coke is reduced or one or two batches of mixture are charged without coke.

3. Excess of quartzite, with the amount of coke being correct, is the cause of acid slags. The slags become cold. The carbon content in the alloy rises, the alloy crusts and freezes in the taphole. The furnace

top becomes dark, electrode positioning is deep. To restore the normal run of the furnace the quantity of quartzite in the mixture is diminished, though an even more drastic method is to exclude quartzite entirely from a number of mixture batches.

4. Lack of quartzite, with the amount of coke being correct, causes the formation of hard-to-fuse and viscous slags with large amounts of entangled metal prills; slag salamanders grow round the electrodes, white smoke evolves at the top; the slag is dense, viscous and is tapped with difficulty. The alloy is overheated. The carbon content in the alloy decreases. Quartzite should be increased in the mixture and a special portion of quartzite (approximately 300 kg) should be charged to eliminate furnace trouble.

5. Lack of both the quartzite and fine coke in the furnace leads to the formation of hard-to-fuse viscous slags which trap considerable amounts of prills of metal and unreduced ore. Slag does not flow out of the furnace, and erodes furnace walls. The silica content in the slag is low, and that of carbon and silicon in the alloy diminishes too.

The positioning of electrodes is high, arcing is audible, blow-outs appear round the electrodes.

Additional fine coke and quartzite should simultaneously be charged into the furnace.

6. Lack of fine coke, with quartzite being in excess, gives birth to cold and viscous slags containing much silica, chrome oxide and iron monoxide. The positioning of electrodes is low, the electrodes are slagged, their consumption goes up. The carbon content in the alloy increases while that of silicon decreases.

In this case it is necessary to check the proportion of the mixture and increase the weight of the reducer in the mixture.

7. Excess of coke and quartzite in the mixture is followed by an increase in the silicon content of the alloy. Slagged coke comes through the taphole, the positioning of electrodes is unstable. Ore without coke and quartzite is to be charged to correct the furnace run.

8. Excess of fine coke, along with lack of quartzite, causes the high positioning of electrodes and numerous blow-outs and eruptions of coke from the crucibles; all this makes furnace-top operation difficult. Slags are hot but viscous, their chrome oxide content is low and it is not always possible to tap them.

After a check-up on mixture proportioning, the weight of the reducer per batch should be diminished.

9. Slag with elevated alumina content. In this case slag becomes foamy and looks cold and acid. There is considerable slagging round the electrodes, their positioning is unstable, the alloy is cold.

To improve the furnace run reverts should not be charged following a check on proportioning and ore quality.

Slag and alloy are tapped into a firebrick-lined ladle simultaneously through the same taphole, three-four times per shift. Slag is poured from the ladle into cast-iron slag pots.

After the tapping the taphole is plugged with refractory clay as deep as possible, for a shallow-plugged taphole may cause damage to the adjacent lining, which in its turn may hamper the operation of the taphole and lead to its breakdown.

Teeming into sectional cast-iron pans is effected by means of a ladle with a sleeve in the bottom.

During the teeming a sample is taken with a spoon and analysed for Cr, Si, C, P and S content. On the following day cooled alloy ingots are smashed into lumps of not more than 15 kg, cleared of slag and stored. Foundry ferrochrome, as a rule, is

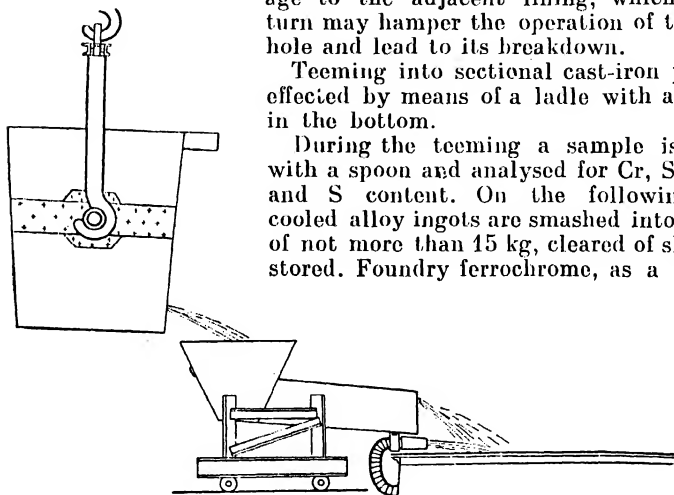


Fig. 73. Diagram of foundry ferrochrome granulation

granulated. A schematic diagram of foundry ferrochrome granulation is shown in Fig. 73.

Consumption of raw materials per one basic ton and chrome recovery in the smelting of foundry and carbon ferrochrome:

	Foundry ferrochrome	Carbon ferrochrome
Chrome ore, kg	1,850	1,911
Its chrome content, kg	663	654
Rich slag, kg	—	100
Its chrome content, kg	—	21
Fine coke, kg	495	470
Quartzite, kg	140	70
Electric power consumption, kWh	3,500	3,500
Anode paste, kg	27	27
Recovery of chrome, %	90.0	89.0

Production of Ferrochrome Silicon

Ferrochrome silicon is used as a reducer in smelting various grades of refined ferrochrome. The carbon content in refined ferrochrome depends to a great extent on the ferrochrome silicon carbon content.

Ferrochrome silicon may also be used in the manufacture of various grades of low-alloy chrome and chrome silicon stainless steels.

Ferrochrome silicon is an alloy of silicon, chrome and iron, with a negligible amount of carbon.

Carbon and silicon contents in ferrochrome silicon are correlated (see Fig. 74). The diagram shows that the higher the silicon content in the alloy the lower its carbon content.

There are two grades of ferrochrome silicon: "rich", with 40-55% Si, and "poor", with 27-34% Si.

Rich ferrochrome silicon, with about 50% of silicon and less than 0.02% of carbon, is used for the manufacture of grade Xp0000 extra-low carbon ferrochrome.

Ferrochrome silicon with 40-50% of silicon and up to 0.10% of carbon is used for lower grades of extra-low carbon ferrochrome, as well as for the low- and medium-carbon ferrochromes smelted by the flux method.

Rich ferrochrome silicon may be smelted by two methods—slag and slag-free.

In the Soviet Union ferrochrome silicon is smelted by the slag-free method in a continuous process in 7,000-12,500 kVA furnaces at 145-160 V on the low side.

Ferrochrome silicon is smelted in the same way as ferrosilicon. The raw materials are quartzite, fine coke and foundry ferrochrome.

Quartzite and coke specifications are the same as in ferrosilicon manufacture.

It is recommended to granulate foundry ferrochrome as in this form it contains a lesser amount of fines and slag inclusions and no big lumps, which are hard to refine; the silicon content should

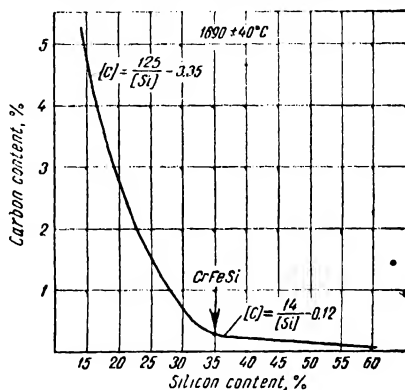
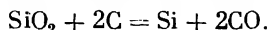


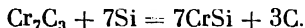
Fig. 74. Relationship between carbon solubility in liquid ferrochrome and the silicon content

not be below 2.5%. The higher the silicon content in foundry ferrochrome the lower is power consumption in ferrochrome silicon smelting.

Silica is reduced by the summary reaction



Reduced silicon dissolves in foundry ferrochrome, displacing carbon and destroying chrome carbides according to the reaction



One of the plants smelting ferrochrome silicon uses a batch of the following composition, kg:

Quartzite	350
Fine coke	170-180
Foundry ferrochrome	150-160
Iron chips	40

Technological operations at the furnace-top in ferrochrome silicon smelting in principle do not differ from those in ferrochrome smelting.

The ferrochrome silicon smelting process calls for uniform evolution of gases over the whole furnace-top area.

This has been attained at the Chelyabinsk Ferroalloy Smelter by introducing sawdust into the mixture to promote its porosity and the deep positioning of electrodes (1,000-1,200 mm). The latter factor ensures normal furnace operation and precludes downsides of mixture round the electrodes. It should be stressed that mixture collapses in ferrochrome silicon smelting are absolutely intolerable; they cause a sharp increase of the carbon content in the alloy because the lumps of foundry ferrochrome which slide down do not always have the time to be refined.

The alloy is tapped four-five times per shift in a thoroughly dried ladle lined with firebrick. The tapping takes 15-25 minutes. The taphole should be opened wide to allow the fullest possible evacuation of slag, since its accumulation in the furnace leads to operation troubles and taphole breakdowns.

To obtain ferrochrome silicon with 0.02% of carbon the alloy should be kept in the ladle for 25-40 minutes before it is teemed to allow the carbon (dissolved in the alloy) to segregate in the form of carborundum and its entangled particles rise to surface; the slag crust, which contains an elevated amount of carbon, should then be removed from the ladle.

After the killing in the ladle the alloy is granulated or teemed into ingots.

Granulation is to be preferred, for the operation may be fully mechanised, and that minimises alloy contamination by carbon-bearing impurities because the remaining slag is washed away by water jets.

Moreover, granulation eliminates the unavoidable ingot-crushing losses. It should be done cautiously, for a granulation rate upwards of 3-4 kg/sec may lead to explosions.

Changes in carbon content in the ferrochrome silicon, following a 20-minute killing in the ladle and granulation, are shown below:

Carbon content

at tapping, % . . . 0.17 0.14 0.12 0.07 0.07 0.07 0.16 0.13 0.12 0.04

Carbon content

following gran-

ulation, % Traces 0.01 0.01 0.01 0.02 Traces Traces 0.02 0.09 0.02

The chemical composition of commercial ferrochrome silicon heats is given in Table 33.

Table 33

Element	Content, %		
Si	48.6	50.4	52.6
C	0.04	0.02	0.01
Cr	33.46	30.70	29.64
Fe	20.8	18.8	17.7

The composition of the slag formed in the production of ferrochrome silicon is given in Table 34.

Table 34

Component	Content, %		
SiO ₂	42.42	58.44	51.36
Cr ₂ O ₃	0.56	3.50	1.02
FeO	1.80	2.06	1.50
Al ₂ O ₃	24.17	14.01	25.28
MgO	0.71	2.05	2.50
SiC	3.60	14.00	10.50
CaO	16.74	5.94	7.82

Every ferrochrome silicon heat is checked for silicon and carbon content, while chrome content is determined by a shift analysis.

Following granulation the ladle is cleared of slag and crusts. The slag is dumped while the crusts are resmelted. Raw-material,

electrode and power consumption and chrome recovery in ferrochrome silicon smelting are given in Table 35.

Table 35

Raw-Material and Power Consumption Per Ton of Ferrochrome Silicon and Recovery of Chrome

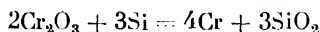
Material	Silicon content, %	
	30	50
Quartzite, kg	650	1,100
Foundry ferrochrome, kg	750	530
Iron chips, kg	35	35
Fine coke, kg	300	500
Power, kWh	3,200	5,600
Anode paste, kg	16	28
Recovery of chrome, %	92	95

Production of Medium-Carbon Ferrochrome

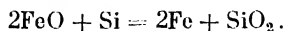
There are two methods of manufacturing medium-carbon ferrochrome: refining carbon ferrochrome by means of chrome ore and refining ferrochrome silicon by means of chrome ore. The latter method, called electrosilicothermic, is employed in the Soviet Union.

In this case, the silicon of ferrochrome silicon reduces (in an electric furnace) the chrome and iron oxides contained in the chrome ore.

The manufacture of refined ferrochrome by the electrosilicothermic method proceeds according to the following basic reactions:



and



Moreover, the chrome contained in ferrochrome silicon is reverted to the alloy.

Silica (SiO_2) is formed through the reduction of chrome and iron oxides by silicon; it accumulates in the slag, thus hampering further chrome oxide reduction.

In the electrosilicothermic method lime is added to the melt to decrease chrome losses (as Cr_2O_3); the calcium oxide contained in lime bonds SiO_2 into calcium silicates $\text{CaO} \cdot \text{SiO}_2$ and $2\text{CaO} \cdot \text{SiO}_2$, thus facilitating chrome oxide reduction in the melt.

Until recently, medium-carbon ferrochrome was smelted by the flux-free method based on the refining of poor ferrochrome silicon (containing 27-34% Si and 40-46% Cr) by chrome ore without lime. The rich slag (27-35% Cr_2O_3) thus obtained was used for smelting foundry and carbon ferrochrome.

Although this method had certain advantages—high yield, no lime required, low power consumption—it had to be abandoned because the smelters could not process all the rich-in-chrome slag produce.

Being cheap, it may be used in the future, provided all the slag obtained is utilised; the alloy produced by this method is low in phosphorus (since lime—the main carrier of phosphorus—is excluded from the process) while the slag it produces is extra-low in phosphorus and is a valuable product for the manufacture of foundry ferrochrome.

Medium-carbon ferrochrome is currently produced by the flux method in electric furnaces with a line voltage of 300 V.

The mixture is composed of chrome ore, lime, ferrochrome silicon and foundry ferrochrome, the latter being introduced to raise the carbon content in the alloy.

Ore is crushed into lumps not larger than 80 mm; lime should contain at least 90% CaO and not more than 0.05% P.

Medium-carbon ferrochrome is smelted by the flux method in three stages:

- 1) charging and melt-down;
- 2) charging of the second batch, melt-down and slagging off through a lap hole into a ladle;
- 3) melt-down of the third batch and alloy tapping through the metal taphole into a slag-filled ladle.

The composition of the mixture used by one of the plants is as follows, kg:

Aktyubinsk ore	1,980
Lime	1,920-2,100
50% ferrochrome silicon	750-810
Foundry ferrochrome	140-160

Electric power consumption for the melt-down of the mixtures should be as follows, kWh:

First batch	2,800
Second "	3,300
Third "	3,600

Total per heat 9,700

The smelting process for medium-carbon ferrochrome by the flux method is similar to that of extra-low carbon and low-carbon ferrochrome which will be described below.

Three slag tappings and three metal tappings are carried out during a shift. The alloy is teemed into whitened cast-iron pans. The ingots are removed from the pan (while still hot) into special containers, cooled and then conveyed to the finished-products storage area where, after an analysis, they are dressed, weighed and shipped to consumers.

Raw-material and power consumption per ton of alloy and chrome recovery by the flux and flux-free methods are given in Table 36.

Table 36

**Raw-Material and Power Consumption and Recovery
of Chrome in the Smelting of Medium-Carbon Ferrochrome**

Materials	Specific consumption of mixture components, kg/ton	
	flux method	flux-free method
Chrome ore (50% Cr_2O_3)	1,350	1,940
Silicochrome (50% Si)	550	—
" (30% Si)	—	750
Lime	1,300	—
Foundry ferrochrome (60% Cr) . . .	130	—
Anode paste	12	25
Iron rods	4.5	2.5
Power	2,220 kWh	1,800 kWh
Recovery of chrome	83%	62%

Production of Low-Carbon and Extra-Low Carbon Ferrochrome

Chrome ore from the Don deposit, ferrochrome silicon and lime constitute the mixture for the production of low-carbon and extra-low carbon ferrochrome.

Chrome ore should be up to specifications set for grades ДХ-1 and ДХ-2. Ore is fed into the furnace in lumps maximum 80 mm.

Ferrochrome silicon with up to 0.02% C and approximately 50% Si is used for the smelting of grade Xp0000 ferrochrome; the ferrochrome silicon used for the smelting of grade Xp000 ferrochrome should contain up to 0.02% C and 46.0-49.9% Si or 0.03-0.05% C and 50-55% Si.

Ferrochrome silicon with up to 0.10% C and 46-50% Si is used for the smelting of Xp00, Xp0, Xp01 ferrochrome.

The lime should be freshly burned and contain at least 90% CaO and maximum 0.025% P. It is crushed into lumps not larger than 80 mm.

Satisfactory results are obtained with the use of lime calcined in tubular rotary kilns fired with gas (Fig. 75).

This lime is not contaminated by carbon-bearing materials in the calcination process and is uniform in size; its calcium oxide content exceeds 93%.

Raw materials should be carbon-free. The composition of the mixture is set according to calculations and is checked in the process of smelting. A simplified charge estimate for the smelting of grade Xp0000 ferrochrome is given below.

The charge is calculated for 100 kg of chrome ore. The chrome ore assays: 53% Cr_2O_3 , 14% FeO, 7% SiO_2 , 12% MgO, 11% Al_2O_3 , 2% calcination losses, 1% moisture.

Ferrochrome silicon—the reducer—contains 52% Si, 31% Cr, 0.02% C and 16.98% Fe.

Lime with 90% CaO is used as a flux.

Chrome recovery from ore is assumed to be 82%, while that of iron amounts to 90%.

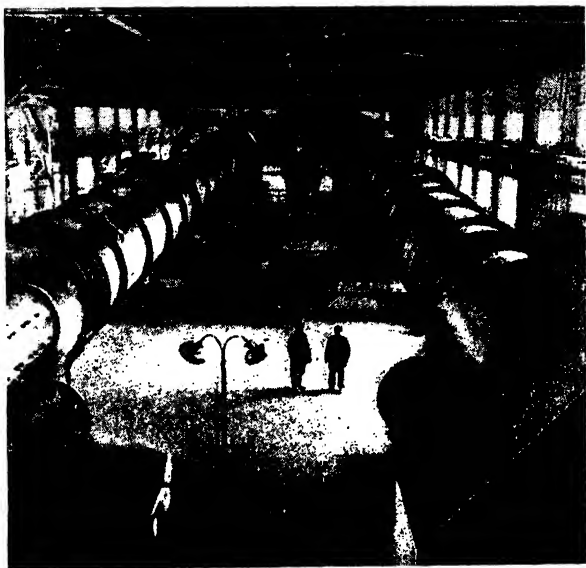


Fig. 75. Rotating kilns for the calcination of limestone

Silicon efficiency in chrome and iron oxide reduction from ore is assumed to be 70%.

Slag basicity, i.e., the $\frac{\text{CaO}+\text{MgO}}{\text{SiO}_2}$ ratio is assumed to be 2.2.

The charge is calculated so as to obtain an alloy containing 0.06% C and 0.70% Si.

The following amounts of components per 100 kg of chrome ore are reduced and reverted to alloy:

chrome

$$100 \times 0.53 \times \frac{104}{152} \times 0.82 = 29.73 \text{ kg};$$

iron

$$100 \times 0.14 \times \frac{56}{72} \times 0.9 = 9.8 \text{ kg}.$$

Amount of silicon required for the reduction of chrome oxide by the reaction $2\text{Cr}_2\text{O}_3 + 3\text{Si} = 4\text{Cr} + 3\text{SiO}_2$:

$$29.73 \times \frac{84}{208} = 12.06 \text{ kg};$$

iron monoxide by the reaction $2\text{FeO} + \text{Si} = 2\text{Fe} + \text{SiO}_2$:

$$9.8 \times \frac{28}{112} = 2.51 \text{ kg}.$$

Total amount of silicon required for the reduction:

$$12.06 + 2.51 = 14.57 \text{ kg}.$$

Amount of ferrochrome silicon required, volatilisation losses included:

$$\frac{14.57}{0.52 \times 0.70} = 40.03 \text{ kg}.$$

The following amounts of components are reverted to the alloy from ferrochrome silicon:

chrome

$$40.03 \times 0.3100 = 12.41 \text{ kg};$$

iron

$$40.03 \times 0.1698 = 6.80 \text{ kg};$$

carbon

$$40.03 \times 0.0002 = 0.008 \text{ kg}.$$

The remaining amount of carbon (up to the 0.06% level) is reverted to the alloy from the electrodes.

Let us calculate the weight and composition of the alloy (Table 37).

Table 37

Element	Amount	
	kg	%
Chrome	$29.73 + 12.41 = 42.14$	71.09
Iron	$9.8 + 6.80 = 16.60$	28.05
Silicon	0.42	0.70
Carbon	0.03	0.06
Total	59.19	100.00

The following amount of silica resulting from the oxidation of silicon contained in ferrochrome silicon is reverted to the slag:

$$(40.03 \times 0.52 - 0.42) \times \frac{60}{28} = 43.70 \text{ kg.}$$

The following amount of silica is reverted to the slag from chrome ore:

$$100 \times 0.07 = 7.00 \text{ kg.}$$

Total silica content in the slag:

$$43.70 + 7.00 = 50.70 \text{ kg.}$$

Amount of basic oxides required (CaO + MgO) in the slag:

$$50.70 \times 2.2 = 111.54 \text{ kg.}$$

Amount of MgO reverted to slag from chrome ore:

$$100 \times 0.12 = 12.00 \text{ kg.}$$

Amount of CaO to be introduced by lime:

$$111.54 - 12.00 = 99.54 \text{ kg;}$$

this requires $\frac{99.54}{0.90} = 110.6 \text{ kg of lime.}$

The weight and composition of the slag are given in Table 38.

Table 38

Constituent	Amount	
	kg	%
SiO ₂	50.70	27.527
CaO	99.54	54.045
MgO	12.00	6.515
Al ₂ O ₃	11.00	5.973
Cr ₂ O ₃	$100 \times 0.53 \times 0.18 = 9.54$	5.180
FeO	$100 \times 0.14 \times 0.10 = 1.40$	0.760
Total	184.18	100.000

Slag to alloy ratio:

$$\frac{184.18}{59.19} = 3.11.$$

Composition of the mixture, kg:

Chrome ore	100.00
Ferrochrome silicon	40.03
Lime	110.60

Chrome content in the mixture:

$$100 \times 0.53 \times \frac{104}{152} + 12.41 = 36.26 + 12.41 = 48.67 \text{ kg.}$$

Total chrome recovery to the alloy:

$$\frac{42.14}{48.67} \times 100 = 86.5 \text{ \%}.$$

Furnaces smelting low-carbon and extra-low carbon ferrochrome are lined with magnesite firebrick. Furnace power rating is 2,500-3,500 kVA.

Extra-low carbon ferrochrome is smelted with a secondary voltage of 300-350 V to ensure lower carburisation of the alloy by the electrodes.

Electrodes used are of graphite or self-baking type.

When low-carbon ferrochrome is smelted, ore and lime are charged simultaneously with ferrochrome silicon, i.e., the reduction of chrome and iron oxides proceeds simultaneously with the melt-down; this shortens the duration of the heat through improved utilisation of the heat generated by chrome and iron oxides reduction reactions.

In smelting extra-low carbon ferrochrome, ferrochrome silicon is charged following the melt-down of the previously charged ore and lime; this ensures a decrease in the carbon content in the alloy (because the furnace operates on highly conductive lime slags without silica additions) and reduces chrome losses in the form of metal prills entangled in the slag.

The smelting of extra-low carbon ferrochrome is a periodic process and consists of the following stages:

- 1) charging of the burden (ore and lime);
- 2) melt-down of the ore-lime mixture;
- 3) reduction of the ore-lime melt by ferrochrome silicon;
- 4) tapping of the heat.

After the melt-down of two charge batches slag is tapped into a ladle; after the melt-down of the third batch slag and alloy are tapped into a ladle filled with slag. The technology described requires two tapholes: a slaghole through which slag is tapped, and a taphole for metal through which alloy and slag are evacuated.

In some smelters extra-low carbon ferrochrome is smelted by the following method. The furnace is charged with 1,800 kg of ore and the estimated amount of lime. At the beginning of the melt-down (8-10 minutes) the electrodes are adjusted manually; after the formation of fused holes round the electrodes, the furnace is switched over to automatic control.

The melt-down of the first charge ends with the consumption of approximately 1,600 kWh of power; following this, the remaining unsmelted mixture is pushed away from the walls, the bath is heated, the slag crust between the electrodes is smashed and ferrochrome silicon is charged when 1,800 kWh of power have been consumed.

The charging of ferrochrome silicon into the bath takes three to five minutes. When 2,000-2,200 kWh have been consumed the melting of the first charge is completed and the electrodes are then lifted for two or three minutes to permit the formation of the slag crust. If this is not done, the slag starts boiling violently when the second batch of the mixture is dumped.

At the same time, the mixture remaining on the banks from the first charge is pushed towards the electrodes. Power input is raised directly on the slag and the second batch is charged into the furnace.

Operations at the furnace-top during the melting of the second batch are the same as during the first.

The second batch of ferrochrome silicon is charged into the furnace when power consumption from the beginning of the heat reaches approximately 4,800 kWh. The heat is considered completed when 6,400-6,600 kWh have been consumed.

The alloy is sampled for its silicon content ten minutes before the end of the heat; the alloy and slag are then tapped into steel ladles.

After the tapping of the alloy the hearth is inspected, the crust is knocked down from furnace rims and the walls are fettled with the mixture left over from the preceding heat.

Prior to applying power it is necessary to charge 150-200 kg of lime and 100-150 kg of ferrochrome silicon in successive layers beneath the electrodes on the skin of the slag remaining from the heat. All this is done to prevent carburisation of the alloy when furnace power is reapplied, especially when grade Xp0000 ferrochrome is smelted.

Power input to the electrodes should be raised very carefully; electrode penetration into the slag and its contact with the alloy remaining in the furnace should be avoided.

The mixture is to be charged when power input reaches 25% of the nominal rating. The mixture is charged into the furnace from the furnace bins mainly near the walls.

The process is conducted with short electric arcs whose length is adjusted by pushing the unsmelted mixture into the holes round

the electrodes. The mixture is fed by small portions as the holes appear at the electrodes; the rate at which the mixture is fed should ensure stable and nominal power input and prevent electrode penetration into the slag.

When the charge collapses and slag agitation sets in, the electrodes should be lifted temporarily to decrease power input.

The initial charge following the tapping of metal is melted incompletely; ferrochrome silicon is charged into the holes at the electrodes; after the consumption of the amount of power preset for the initial charge the electrodes are lifted and, as soon as the slag crusts, power input is again raised and a second mixture batch is dumped. In the case of 1,800 kg of ore and the estimated amount of lime, power consumption for the first charge batch usually comes to 2,100-2,400 kWh.

The second and third batches are melted down completely; if the furnace operates normally power consumption amounts to 160 kWh per 100 kg of ore. The mixture should be melted down entirely with no unsmelted materials remaining between the electrodes and the furnace sides. The slag crust between the electrodes is broken up before ferrochrome silicon is fed into the bath after the melt-down of the ore-lime portion of the mixture. The charging of ferrochrome silicon takes ten minutes.

If the slag does not boil when ferrochrome silicon is being charged and the reducer remains at the surface of the slag, the bath is mixed artificially with an addition of two or three shovelfuls of chrome ore (to provoke bath agitation).

Ferrochrome silicon must not be charged into the electric arc zone, for that causes useless volatilisation of silicon.

After the entire portion of the reducer has been charged the bath is mixed once more, the furnace is operated for ten minutes and the slag and alloy are then tapped. Prior to tapping of the alloy three samples are taken by a spoon and the silicon content is assayed visually. Electric energy consumption for the smelting of the second and third batches, with the ore weighing 1,800 kg per charge, averages 3,000 and 3,000-3,300 kWh respectively.

If the alloy has an elevated silicon content the tapping is held over, the heat is prolonged and the alloy is additionally refined (lime and chrome ore are added to the melt). If this does not suffice, the slaghole is opened, the slag is tapped and a fourth batch (with the amount of lime increased by 200-300 kg against the normal quota) is charged. When this charge has been melted a diminished amount of ferrochrome silicon (but not less than $\frac{2}{3}$ of the normal amount so as to prevent "cold" furnace run) is introduced. The alloy bath is then sampled and, if the alloy is up to standard specifications, the heat is tapped out.

A so-called cold furnace run may come about in manufacturing low- and extra-low carbon ferrochrome; this occurrence is characterized by elevated power consumption and a high silicon content in the alloy. Cold furnace operation may be caused by:

- a) inadequate top operation which may lead to local overheating of the slag, long arc operation and high heat losses;
- b) low ferrochrome silicon content in the mixture;
- c) low calcium oxide content in the lime which increases power consumption for its final burning in the furnace;
- d) prolonged furnace down-time;
- e) excessive amounts of ferrochrome scrap being remelted.

When the carbon content in the alloy rises, technological operations at the furnace-top should be activated, raw materials carefully checked for carbon contamination and electrode tips inspected for crumbling.

If the slag analysis reveals a low calcium oxide content (below 48%) or an increased chrome oxide content (above 5%), one should check on the correctness of the proportion, taking into account actual raw-material analyses; if necessary, the proportion of lime and ferrochrome silicon should be modified.

As mentioned above, when the second mixture batch has been melted, the slaghole is opened and the slag is tapped into a steel cast ladle. The latter should be filled with slag to the lip. Following the tapping of the slag its surface in the ladle is covered with slag dust 50 to 60 mm thick to prevent the formation of a thick slag crust. Prior to the tapping slag crust in the ladle is smashed and when the slag and alloy are tapped the latter washes the liquid slag away and fills the ladle.

The taphole is opened either with a bar, or with an electric arc aided by an iron rod or with oxygen lancing. To tap the slag and alloy the taphole is opened wide to permit their simultaneous outflow and thus reduce tapping time to ten minutes or less. Excessive slag flows over from the first ladle into the second, then into the third or into cast-iron slag pots.

After the tapping procedure the taphole is closed with magnesite powder as deep as possible. Deep taphole plugging enhances front-wall heat resistance and facilitates subsequent tapping.

The ladle with the metal is then conveyed to the pouring side where the alloy is teemed.

When the ladle is not filled with the alloy completely, the slag is poured out into slag pots; the small amount of slag remaining in the ladle is thickened by an addition of crushed lime to prevent slag from infiltrating into the teemed alloy.

Ferrochrome is teemed into whitened steel or cast-iron moulds, into ingots 50 to 100 mm thick.

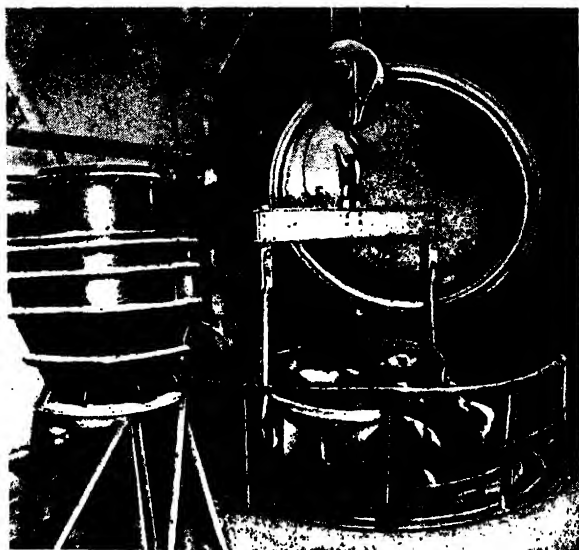


Fig. 76. Installation for vacuum-processing liquid of ferrochrome in a ladle

At some plants the ladle with the alloy is vacuum-refined prior to teeming. This permits obtaining a dense ferrochrome ingot. A vacuum-refining installation is shown in Fig. 76. After solidification ferrochrome ingots are stripped and sent to the finished-products' area where they are crushed into lumps weighing not more than 15 kg, cleared of slag, packed into cases (on the batch principle) and shipped to consumers. In some instances the alloy is shipped to consumers in bulk (without packing).

Every heat is sampled and analysed for chromium, silicon, carbon and phosphorus content.

The approximate chemical composition of extra-low carbon ferrochrome is as follows: 70-71% Cr, 0.06-0.08% C, 0.60-0.80% Si, 0.042-0.051% P.

The correct proportion of reducer and lime is checked by shift analyses of slag for Cr_2O_3 , CaO and SiO_2 content.

Approximate slag composition: 51-52% CaO, 3.8-5.4% Cr_2O_3 , 5.3-6.5% Al_2O_3 , 0.6-0.8% FeO, 8.5-10.0% MgO, 26-29% SiO_2 .

Consumption of raw materials, electrodes and power, as well as chrome recovery in extra-low carbon ferrochrome smelting are given below:

Materials

Chrome ore (50% Cr_2O_3), kg/ton	1,600
Chrome silicon (50% Si), kg/ton	600
Lime, kg/ton	1,350
Electrodes, graphitised, kg/ton	10
Iron rods, kg/ton	3.5
Electric power, kWh/ton	2,300
Chrome recovery, %	82

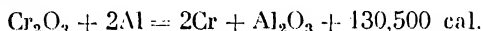
Special installations have been put up in the Soviet smelters manufacturing ferrochrome to recover scrap and ferrochrome prills from the slag. In new smelters special facilities are provided to process the slag and recover alloy prills and to use disintegrating slag as a building material.

Aluminothermic Production of Metallic Chrome and Special Chromium Alloys

The aluminothermic method is used for the production of certain grades of ferrochrome, metallic chrome and alloys.

Tables 30 and 31 give the composition of metallic chrome and certain of its alloys manufactured in the Soviet Union by the aluminothermic method.

The reduction of chrome oxide by aluminium proceeds according to the reaction

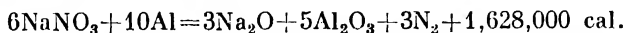


If we divide the heat effect of the reactions by the sum of the atomic weights of Cr_2O_3 and 2Al ($2 \times 52 + 3 \times 16 + 2 \times 27 = 206$) we see that the amount of heat generated per 1 kg of mixture is equal to

$$\frac{130,500}{206} = 633 \text{ cal.}$$

This is more than required according to the rule set by Zhemchuzhny, who arrived at the conclusion that the realisation of the aluminothermic process outside a furnace requires the specific heat effect, i.e., the amount of that evolving in the reduction process per kg of mixture, to be above 550 cal.

Slags formed in the aluminothermic process are hard to fuse and an increase of up to 620-680 cal in specific heat effect is therefore necessary; this is achieved by the introduction into the mixture of sodium nitrate, which, reacting with aluminium, generates a considerable amount of heat:



Pre-heating of raw materials is worthwhile from the economic point of view.

Recovery coefficients for charge calculations (based on operational practice) are given in Table 39.

Table 39

Element	Reverts to the alloy, %			
	metallic chrome	extra-low carbon ferrochrome	nitrogenated ferrochrome	XAЛ
Chrome	93	87	85.0	97
Chrome in prills included	5	15	15.0	15
Silicon	40	40	40.0	40
Iron	100	100	100	100
Specific heat effect of the process, cal/kg	655-685	650-680	630-640	630-650
Temperature of the process, °C	2,000-2,100	1,970-2,060	—	—

The upper values of the specific heats of the processes refer to winter conditions, the lower ones to summer conditions.

The estimated charge composition for the aluminothermic smelting of chrome and its alloys is given in Table 40.

Table 40

Alloy	Mixture composition, kg				
	chrome oxide	chrome concentrate	aluminum powder	sodium nitrate	slag from metallic chrome smelting
Metallic chrome	100	—	38.9	8.97	—
Extra-low carbon ferrochrome	—	100.0	31.7	16.5	—
Nitrogenated ferrochrome	—	100.0	39.6	25.0	41.5
XAЛ	—	100.0	46.9	15.8	—

The chemical composition of chrome oxide used in the mixture (grades OX-B and OX-Э), as specified by ГОСТ 2912-54, is shown in Table 41.

Table 41

Grade	Chrome oxide, above %	Total sulphur, below %	Iron monoxide, below %	Moisture, below %
OX-B	98.0	0.06	—	0.15
OX-Э	98.0	0.03	0.15	0.15

Outwardly, chrome oxide is a fine, homogeneous powder, green in colour.

Chrome concentrate should contain not less than 60% Cr_2O_3 , not more than 15% FeO , 1.2% SiO_2 and 0.1% C. Chrome concentrate grains should not exceed 1.5 mm.

The chrome slag obtained in the smelting of metallic chrome contains approximately 12% Cr_2O_3 . It is crushed and sieved through a 1.5-mm sieve and its sizing assay is as follows:

more than 0.8 mm	0.3-0.8 mm	less than 0.3 mm
20-30%	20-40%	30-60%

Aluminium powder is manufactured from primary aluminium whose chemical composition is specified by ГОСТ 3549-47 (Table 42); its sieved analysis should be as follows:

Size, mm	Up to 0.1	0.1-1.0	1.0-3.0
Content, %	<10.0	>80.0	<10.0

Table 42

Chemical Composition of Primary Aluminium

Grade	Al, minimum	Chemical composition, %				
		Fe	Si	Sum Fe+Si	Cu	Sum of impurities
		maximum				
A-00	99.7	0.16	0.16	0.26	0.01	0.30
A-0	99.6	0.25	0.20	0.36	0.01	0.40
A-1	99.5	0.30	0.30	0.45	0.015	0.50

Sodium nitrate should be freshly dried and contain at least 98% NaNO_3 .

The amount of chrome oxide or chrome concentrate smelted per heat ranges from 1,000 to 3,000 kg. The burden is carefully mixed for 30 minutes in a drum mixer.

The smelting is done in a sectional cast-iron shaft furnace. The furnace is prepared in the following manner. Prior to its assembly the inside of the shaft is whitened with lime; the shaft halves are then installed and bolted together on a carriage lined with firebrick. The joints of the halves are luted with fireclay. Dry magnesite pow-

der is spread over the bottom and tamped to a layer 150 mm thick. The lower section of the shaft, which is filled with liquid metal during the operation, is lined with dry magnesite brick, all joints being packed with magnesite powder. The joints of the halves are also protected with firebrick. The bottom is then covered with a 30-mm layer of slag, milled and sieved through a 3-mm sieve.

Metallic chrome and extra-low carbon ferrochrome are smelted with bottom priming which ensures a uniform process. The mixture is charged on the shaft bottom up to the magnesite lining level before the primer is inserted. The primer mixture is put into a hole situated in the charge (0.5 kg magnesium chips and 0.5 kg saltpetre). The mixture is charged after the ignition of the primer and as soon as the process is initiated. The mixture is spread in a thin layer all over the charge surface. It is inadvisable to operate with an open top as this leads to heat losses; it is not recommended to charge a thick layer of saltpetre over the top either, for that causes decomposition of saltpetre by the heat of the furnace-top (evolution of brownish gases). The decomposition of the saltpetre by furnace-top heat increases the aluminium content in the metal. The mixture, especially when one is smelting ferrochrome, should be charged in a manner excluding any interruptions for that may cause burden and melt eruptions by evolving gases.

The process runs hot. The over-all duration of the heat is approximately five minutes.

Nitrogenated chrome and chrome-aluminium alloy are smelted with surface priming, i.e., all the burden is charged into the smelting shaft and the mixture is ignited from the top by means of a primer. The process is very hot and lasts about two minutes.

To facilitate nitrogen assimilation by the alloy the smelting temperature is driven down by the addition of a certain amount of slag from the preceding heats.

The alloy and slag block is cooled for five-six hours following the smelting process; the shaft is then disassembled and dismantled. The alloy and slag monolith is left over on the car for another three-four hours after which the slag is separated from the alloy.

The alloy is water-cooled to facilitate crushing and dressing, to that end it is placed in a tank which is gradually filled with water.

After it has been water-cooled the alloy is shattered to pieces, cleaned and packed into tare.

At the same time the alloy is sampled for its grade. Raw-material consumption per basic ton and chrome recovery in the smelting of these alloys are given in Table 43.

Typical composition of commercial alloy is shown in Table 44.

Table 43

**Raw-Material Consumption Per Basic Ton of Alloy
and Recovery of Chrome**

Alloy	Raw-material consumption, kg				Chrome recovery, %
	Chrome concentrate (60% Cr ₂ O ₃)	Chrome oxide	Aluminum	Sodium nitrate	
Metallic chrome (97% Cr)	—	1,660	630	130	87-88
Extra-low carbon ferrochrome (60% Cr)	2,060	—	650	36.5	68-70
Nitrogenated ferrochrome (60% Cr)	2,150	—	840	540	66-69

Table 44

Commercial Alloy Composition, %

Alloy	Cr	Fe	Si	Al	C	S	P	N
Metallic chrome	98.6	0.65	0.17	0.25	0.04	0.02	0.005	—
	98.8	0.71	0.15	0.31	0.03	0.03	0.007	—
	98.4	0.72	0.14	0.57	0.03	0.01	0.004	—
Extra-low carbon ferrochrome	74.5	—	0.42	0.15	0.03	0.006	—	—
	73.75	—	0.41	0.22	0.02	0.005	—	—
Nitrogenated ferrochrome	73.12	—	0.27	0.40	0.03	0.009	—	1.30
	72.6	—	0.58	0.15	0.04	0.007	—	1.25

Production of Refined Ferrochrome by Other Methods

A new method of extra-low carbon ferrochrome production has been developed and introduced into smelting practice—decarburation of carbon ferrochrome in solid state in vacuum furnaces at 1400-1300°C.

The process is based on the oxidation of ferrochrome carbon by the oxygen of iron, chrome, silicon, etc., oxides by the reaction

$$(\text{Fe, Cr})_7\text{C}_3(\text{solid}) + 3\text{SiO}_2(\text{solid}) = 7(\text{Fe, Cr})_{(\text{solid})} + 3\text{SiO}_{(\text{gas})} + 3\text{CO}_{(\text{gas})}$$

or



The gaseous products SiO and CO resulting from the reaction are pumped out. Deep vacuum (residual pressure is approximately equal to 0.05 mm mercury gauge) ensures the shifting of the system equi-

librium to the right, i.e., facilitates ferrochrome decarburation. This method makes it possible to produce ferrochrome with 0.01-0.02% C.

The process is conducted in electric resistance furnaces at 1100-1300°C; the briquettes prepared from finely ground carbon ferrochrome and reducer are aged for 24 hours in deep vacuum. Chromic anhydride is used as a binder.

The composition of decarburised briquettes depends on the reducer. The results of investigations by I.D. Kirichenko into the oxidiser effect on the composition of decarburised briquettes are given in Table 45.

Table 45

Relationship Between the Composition of Decarburised Briquettes and the Kind of Oxidiser

Oxidiser	Alloy composition, %			
	Cr	Si	Fe	C
Chrome oxide	82.0	0.22	18.3	0.02-0.04
Sand	70.5	7.5-8.5*	21.2	0.02-0.04
Iron ore	63.2	0.70	36.3	0.02-0.04

* Partly in the form of silica.

The advantages of this method of ferrochrome production are considerable improvement in working conditions compared with the silicothermic process of extra-low carbon ferrochrome manufac-

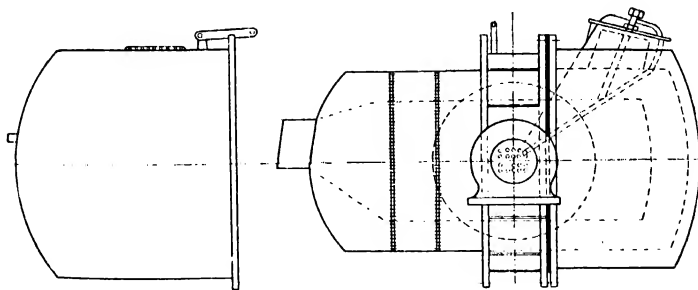


Fig. 77. Diagram of a converter for oxygen blowing of foundry ferrochrome

ture and the possibility of mechanising and automating production. The alloy produced contains 0.02-0.04% C. A serious disadvantage of this method is considerable contamination by the reducer of the

alloy obtained and its high silicon content when silicon is used as an oxidiser.

The production of refined ferrochrome through the processing of foundry ferrochrome in a converter is advantageous from the economic point of view.

Experiments in foundry ferrochrome converting with the use of oxygen have been conducted in a converter (Fig. 77) composed of two sections: the inner body with the lining and the hood, which creates a vacuum in the converter with the aid of vacuum pumps. The converter in question was lined with brick manufactured with cast magnesite, and was water-cooled. The results of the blowing operations are given in Table 46.

Table 46

Ferrochrome Converting Performance Data

No. of heat	Blow duration, min		Weight of metal charged, tons	Analysis of metal charged, %			Weight of metal produced, tons	Analysis of metal produced, %				Grade of metal produced	Oxygen Consumption per ton of metal charged, m ³ /ton	Chrome recovery, %
	usual	oxy-gen		C	Cr	Si		C	Cr	Si	P			
34	158	138	2.810	5.89	61.0	1.55	1.515	0.56	67.8	0.95	0.03	Xp1	110	60
39	130	166	3.110	6.44	71.4	1.03	1.800	1.13	72.78	1.17	0.02	Xp2	91	50
44	165	150	3.535	5.94	67.14	1.40	1.920	0.45	71.9	0.93	0.03	Xp01	92	60
50	139	134	1.470	6.3	70.69	1.03	0.700	0.17	62.42	1.03	0.04	Xp0	173	42

Similar converters are operating at present and produce grades Xp1 and Xp2 ferrochrome with the following performance results:

Foundry ferrochrome consumption, kg/ton of alloy	1,420
Oxygen consumption, m ³ /ton of alloy	200
Aluminium, kg/ton of alloy	2.5
Chrome recovery, %	70.3

Chapter 5

PRODUCTION OF FERROTUNGSTEN

Use of Ferrotungsten

Tungsten is one of the most important alloying elements in the manufacture of special, tool and certain grades of construction steels. Tungsten increases steel's hardness, tensile strength and elasticity. Steels with a high tungsten content (3.5-19%) are used for the manufacture of high-speed cutting instruments. Cemented carbide alloys containing 90% tungsten, currently widely employed, are manufactured of tungsten carbides.

The chemical composition of ferrotungsten is given in Table 47.

Table 47

Chemical Composition of Ferrotungsten (ГОСТ 4752-55)

Grade	Composition, %											
	W mini- mum	C	Mn	P	S	Si	Cu	As	Sb	Sn	Bi	Pb
		maximum										
B0	80	0.2	0.2	0.03	0.05	0.3	0.10	0.04	0.04	0.05	0.04	0.04
B1	70	0.2	0.2	0.04	0.08	0.4	0.15	0.05	0.08	0.10	0.05	0.05
B2	70	0.4	0.4	0.05	0.10	1.0	0.20	0.08	—	0.15	—	—
B3	65	0.8	0.5	0.10	0.20	1.5	0.30	0.08	—	0.25	—	—

Since some tungsten concentrates have a considerable molybdenum content (2-4.5%), ferrotungsten is smelted together with molybdenum, its composition conforming to ГОСТ 5199-50.

Physicochemical Properties of Tungsten

Tungsten (W) is an element of the sixth group of the D. I. Mendeleev Periodic Table.

The over-all tungsten content in the earth's crust is negligible—0.0009%. Tungsten was discovered by the Swedish scientist K. Scheele in 1781 in calcium tungstate CaWO_4 ,

later called scheelite. In 1783 tungsten was found in a mineral called wolframite (Fe, Mn) WO_4 .

For almost 100 years tungsten was not employed and it was only in the latter half of the 19th century that experiments were made to introduce tungsten into steel to improve the latter's properties. Tungsten output began to rise very swiftly early in the 20th century when it found application in the manufacture of rapid steel.

Pure tungsten is manufactured by the reduction of its oxides by hydrogen. Tungsten is a silvery white metal with the following physicochemical properties:

Atomic weight	183.92
Specific gravity	19.32
Valence	From 2 to 6
Melting point, °C	3400
Boiling point, °C	5927
Hardness, H_B	400

Tungsten oxidises noticeably in the air when heated from 400-500°C and up. The melting point of tungsten alloys goes up with the rise in latter's content. The melting point of the alloy containing 70-80% W is 2000°C. With carbon tungsten forms carbide WC, containing 6.12% C, and W_2C , containing 3.16% C. Carbide inclusions make tungsten brittle. The W_2C melting point is 2750°C. Tungsten gives two compounds with silicon: W_2Si_3 and WSi.

With oxygen tungsten forms a number of oxides: tungsten dioxide WO_2 —a brownish powder with specific gravity of 12.11, melting point of 1227-1327°C and boiling point of 1727°C. Tungsten trioxide WO_3 is the most stable tungsten-oxygen compound, occurring in nature in a number of tungsten ores. WO_3 is yellow in colour, its specific gravity being 7.16 and melting point 1473°C.

Raw Materials

The main raw materials used in the production of ferrotungsten are tungsten ore concentrates. Since the tungsten assay in the ores is very low, they are considered to be commercial if they contain over 0.2% of tungsten trioxide WO_3 .

Wolframite, scheelite, ferberite and hubnerite are the main tungsten minerals.

Wolframite (Fe, Mn) WO_4 is a black brownish mineral with a metallic lustre; its specific gravity is 7.14-7.50. The pure mineral assays 75% WO_3 , 25% $\text{MnO} + \text{FeO}$.

Scheelite CaWO_4 is light-yellow to dark-red in colour, with specific gravity of 5.9-6.1. The WO_3 assay in the mineral varies from 71 to 80%, depending on the amount of impurities.

Ferberite FeWO_4 is a black mineral with specific gravity of 7.5; it contains 76.5% WO_3 and 23.5% FeO .

Hubnerite MnWO_4 is a brown or dark-red mineral with specific gravity of 7.2.

The most common of the above-mentioned minerals are the wolframite ores whose output amounts to 95% of the world's output of tungsten raw materials.

The biggest tungsten ore deposits are to be found in China, Australia, Korea and Burma. The Soviet Union has adequate deposits too. At Soviet works ferrotungsten is manufactured from wolframite and scheelite concentrates, whose chemical composition is given in Table 48.

Table 48

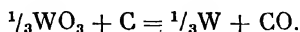
**Chemical Composition of Tungsten Concentrates for Smelting
Ferrotungsten (ГОСТ 213-56)**

Grade	Chemical composition, %											
	WO_3 mini- mum	maximum										
		MnO	SiO_2	P	S	As	Sn	Cu	Pb	Sb	Bi	Mo
КВФШ	63	9	5	0.02	0.3	0.10	0.10	0.07	0.10	0.20	0.50	Not spec- if- ied
КВГ	60	15	5	0.05	0.8	0.20	0.20	0.20	Not specified			"
КШШ	70	1	0.5	0.05	0.5	0.10	0.10	0.10	0.10	0.10	0.10	"
КШО	60	2	10	0.04	0.6	0.05	0.08	0.15	Not specified			"
КШ	55	4	10	0.08	0.8	0.20	0.20	0.20	"	"	"	"
КМША	65	0.10	1.2	0.04	0.3	0.04	0.05	0.10	"	"	"	2-4.5
КМПЭ	60	0.10	5	0.08	0.5	0.05	0.05	0.10	"	"	"	2-4.5

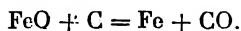
Ferrotungsten Smelting Technology

Oxides of tungsten may be reduced by silicon, aluminium or carbon. The increase in the tungsten content enhances the viscosity of the alloy, thus hampering the latter's carburisation.

The most widely used method of ferrotungsten manufacture is the simultaneous reduction in electric furnaces of tungsten and iron oxides by carbon:



The process begins at 690°C. Iron monoxide is reduced at the same time (beginning with 950°C):



Before 1937 tungsten was smelted in the Soviet Union in the form of ingots.

In 1937 a continuous smelting process developed by V. N. Gusev was introduced into production. It consists of scooping the alloy out of the liquid bath and periodic slag tappings.

The following raw materials are used for the production of ferrotungsten:

1) tungsten concentrate, the tungsten-bearing component of the mixture;

2) coke, used for the reduction of tungsten and iron oxides. The coke used is of the pitch variety, with the minimum content of phosphorus, sulphur and ash. Used are either pitch fines or coke crushed into lumps of less than 20 mm;

3) 75% ferrosilicon is used for rapid and complete reduction of tungsten oxide contained in the slag prior to the latter's tapping. The ferrosilicon used is in granulated form;

4) iron chips introduced into the mixture to reduce the alloy's viscosity (this results in a drop of the tungsten content) make it possible to scoop the alloy. To avoid contamination, the iron chips should contain no non-ferrous metals and cast iron.

The mixture batch is composed of 100 kg of concentrate. The latter is added in proportions which permit obtaining a standard metal in both the principal element and impurities. The distribution of elements in the smelting process is given in Table 49.

Table 49

Distribution of Elements

Element	Reverts, %		
	in metal	in slag	in stack losses
W	99*	0.15	0.80
Mn	4	85	11
Fe	90	7	3
P	100**	—	—
Cu	90	10	—
As	20	—	80
Sn	25	—	75
S	10***	40	50

* 0.05% other impurities.

** Of that contained in the concentrate, assuming that the phosphorus of the remaining mixture component constitutes stack losses.

*** The same as phosphorus.

The amount of reducer per batch is set so as to run the heat on working slag with ~ approx. 10% of tungsten oxide. The rise of the tungsten



Fig. 78. Concentrate-charging machine

oxide content in the slag prevents the impurities from being carried over to the alloy (carbon, silicon, manganese).

The amount of coke per charge is from 8 to 10 kg. The quantity of iron necessary per 100 kg of concentrate is given by the formula

$$\frac{0.286 \times A - (0.7B + 0.23C + D)}{0.90}$$

where

- A—the mean tungsten oxide content in the mixture concentrates;
- B—the mean iron monoxide content in the mixture concentrates;
- C—the amount of ferrosilicon for slag adjustment per 100 kg of concentrates;
- D—the amount of iron introduced by spoons and iron rods.

Ferrosilicon consumption amounts to 7-10 kg per 100 kg of concentrates.

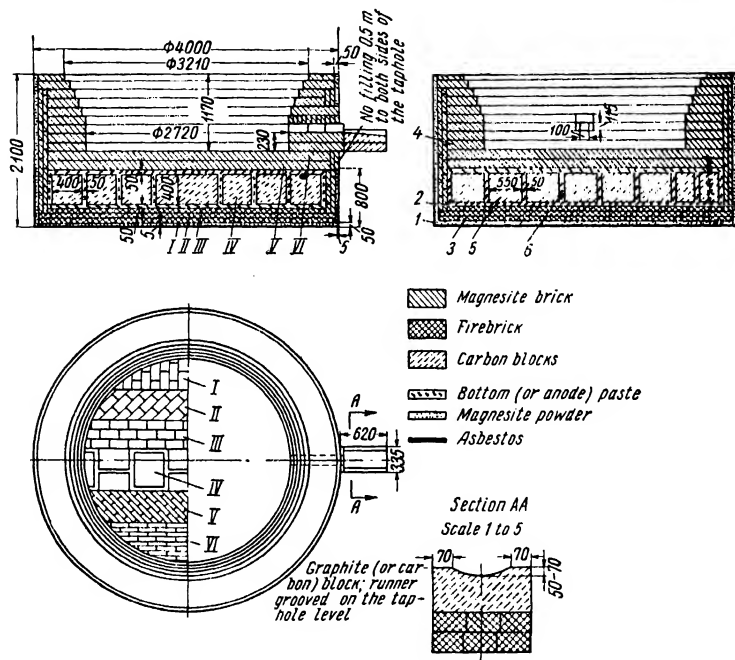
Mixture materials are conveyed from proportioning bins to furnace hoppers in self-unloading containers or charged into a furnace by a machine (Fig. 78).

Ferrotungsten is smelted in a three-phase arc furnace of 2,500 kVA at line voltage of 178 and 149 V.

The furnace is equipped with a rotating mechanism to ensure uniform heating of the metal over the whole furnace hearth.

The lining diagram for a ferrotungsten furnace is shown in Fig. 79.

The creation of a protective metallic layer (of the crucible) is a crucial start-up operation; to this end approximately 10 tons of



ferrotungsten are charged onto the furnace banks. For the first two-three days the alloy is not scooped, and the concentrate is charged simultaneously with coke and the separated slag is tapped out periodically.

The refining of the alloy is effected at 178 V, which speeds up the charging rate and is necessary for slag enrichment. During the scooping-out period the alloy should be paste-like to facilitate this operation; the furnace is, therefore, changed over to the second volt-

age tap during this period (149 V), electric arcs are thus shortened, heat losses decreased and thorough heating of the metal ensured.

The ferrotungsten smelting process consists of the following stages:

Refining	3-3.5 hours
Scooping	1.5-2.5 hours
Slag adjustment	40 minutes
Slag tapping	15-20 minutes
Fettling and electrode slipping	30-40 minutes

The alloy in the furnace is enriched with impurities during the slag-adjustment period (manganese, silicon, carbon). A layer of low-fusing fluid slag, containing 5-7% Si, 2-4% Mn, 52-54% W, is formed on the hearth.

After the fettling of the hearth and electrode slipping, all available reverts and concentrate fines are charged into the furnace. The full amount of iron chips (as calculated for the heat) is charged at the beginning to speed-up refining, with the result that the alloy's viscosity falls along with the decrease of the tungsten content in the latter.

The concentrates are charged in portions of 300-500 kg after the melting-down of iron chips and reverts and the heating of the furnace for 30 minutes. The first batch of concentrates is thoroughly heated. The approximate composition of the slag at that moment is given in Table 50.

Table 50

Approximate Composition of Slag Following the Melt-Down of Iron Chips and Reverts, %

WO ₃	FeO	SiO ₂	MnO	CaO	MgO	Al ₂ O ₃
22.2	19.6	28.5	13.8	9.5	2.7	2.3
35.04	16.3	27.8	12.15	7.6	1.3	2.42
21.20	16.51	32.32	10.8	8.6	1.3	1.8
18.70	20.9	32.4	14.5	10.5	1.2	2.3
24.83	23.1	31.0	8.8	10.1	0.6	1.67

The process of alloy refining from silicon and manganese proceeds at the metal-slag interface. At the beginning, the oxidation of impurities proceeds at high speeds; then, as the silicon and manganese content decreases and the alloy is enriched in tungsten, the process slows down. The speed of the process in this particular case is determined not only by the temperature of the slag and concentrations of reactive components but by the physical properties of the alloy, i.e., by its fluidity.

A hot furnace run and shortening of down-time for fettling and electrode slipping are imperative for speedy alloy refining. The process of active refining usually ends with the intensive boiling of the bath, following which two or three samples are taken to determine the silicon and manganese content (by high-speed analyses). If the metal is up to the specifications, it is scooped out.

Ferrotungsten is scooped out onto the working floor by steel spoons. Spoons with the alloy are cooled in a bath with running water. After that the ingots are knocked out of the spoons and loaded into bins. Control of the quality of the metal during the scooping requires the ingot fracture to be periodically inspected and its quality determined by the following symptoms: a good sample—the lower surface is rough with small hollows and protrusions, the upper surface wavy; the fracture is fine-grained and dull; samples with the high silicon and manganese content have a smooth lustrous lower surface and a silvery fracture with no visible grains. The carbonaceous sample has an iridescent acicular structure and a smooth lower surface.

An alloy poor in tungsten has a smooth upper surface and a macro-crystalline fracture.

During the whole scooping-out period, silicon and manganese contents are checked repeatedly by high-speed analyses. The alloy is scooped from over the whole bath surface.

Normally, approximately 70-75 kg of alloy are scooped per each portion of concentrates charged, this depending on the tungsten oxide content in the concentrates, the protective layer condition and the hearth level. Excessive scooping causes the furnace banks to melt down and may result in wall burn-outs. An important thing in the scooping-out operation is to keep the alloy heated and in paste-like condition, which facilitates this operation. To this end the concentrates are charged by small batches (100-200 kg) during the scooping-out period. Coke is also charged by small portions to keep the normal working slag in slightly foamy condition; the slag contains approximately 11.18% WO_3 , 25.9% FeO , 33.9% SiO_2 , 15.8% MnO , 10.8% CaO , 2.48% MgO , 2.74% Al_2O_3 .

The temperature of the slag may be adjusted by varying the charging rate. The charging rate is normal when power consumption per 100 kg comes to 260-300 kWh; the slag is quite viscous and foamy as a result of CO bubbles. Foamy slag has considerably higher electrical resistance and promotes deep penetration of the electrodes and more efficient heating of the metal. If the charging rate is slowed down or if there is not enough reducer in the mixture, the slag overheats, its foaming decreases or stops entirely, the electrodes are high, the electric arcs are bare, and local hot-spots appear. Besides this, the concentrate charging rate is determined by the condition of the

alloy: if the scooping of the alloy grows difficult the charging rate should be reduced; if the metal is overheated the rate should be increased (coke and concentrates should be fed more frequently). The charging of concentrates is stopped at the end of scooping. Coke is added to the slag by uniform batches to prepare for the adjustment of the slag.

The aim of slag finishing is to drive down its tungsten content prior to tapping. Slag finishing is effected by ferrosilicon, with small amounts of coke being fed at a uniform rate. The degree to which the slag is impoverished is determined by the character of the fractures of the samples taken from the metal bath. Adequately impoverished slag has a green fracture. As soon as a satisfactory sample is obtained the heat is held over for 10-15 minutes to settle the metal prills, and the slag is then tapped. The composition of the final slag is approximately as follows: 0.22% WO_3 , 0.68% FeO , 47.38% SiO_2 , 24.01% MnO , 20.12% CaO .

The taphole is lanced with oxygen or burned through with the aid of the electric tapper. The slag is poured into slag pots through launders fettled with sand.

Twelve samples are taken at equal time intervals during the slag-tapping process to determine the tungsten trioxide content in the mean sample. The slag with WO_3 above 0.3% is resmelted; if it is less, it is dumped.

The hearth is fettled following the draining of the slag. Furnace walls are luted up with semi-liquid metal from the hearth, as well as with metallic waste from ingot dressing and scooping. Following the fettling the electrodes are repositioned. The electrodes are slipped up maximum 400 mm at one go.

The following deviations from normal operating conditions may occur during the smelting process:

1. The metal is high in silicon, manganese or carbon. An increased content of the above-mentioned impurities is a sign that the refining was run cold, its duration was insufficient and that the slag may have been poor at the time.

To drive down silicon, manganese and carbon refining should be continued and scooping stopped. The charging of concentrates should be continued by small batches, and the slag should be sufficiently heated. If the slag is rich in tungsten oxide, no concentrates are to be charged, while the bath is to be adequately heated and the melt systematically poked.

2. The metal is unmelted ("dirty"). The charging of concentrates onto unheated slag, excessive charging rate, etc., may cause the appearance of unmelted metal or unmelted mixture of alloy with concentrates. The formation of a rich, hard-to-fuse layer on the hearth prevents alloy refining and scooping. Further charging of concentrates

in this case enriches the slag and leads to excessive erosion of the furnace sides. Concentrate charging should be discontinued to eliminate the said effect and the hearth should be well heated. In very serious cases iron chips are to be introduced into the furnace.

3. Irregular scooping from the bath surface and the cold run of the furnace (when the alloy melts chiefly underneath the electrodes) may lead to the formation of metallic salamanders—"mushrooms"—which cause furnace trouble. In this case the electrodes become shallow-positioned, the arcs are bare, power input is unstable. Bottom accretions are eliminated by charging iron chips and ferrosilicon, as well as by intensified scooping. Pickling with iron ore or scales may be resorted to when large accretions appear.

4. Operation with very rich slags may bring about the break-down of the bath. As the banks are fettled with a non-refined alloy the considerable amounts of impurities contained therein may react with the rich slags, and, as the result, the protective layer heats up and melts. The following steps should be taken in these circumstances (depending on the degree of erosion):

a) when the erosion of the sides is negligible small amounts of coke should be added to those spots where erosion is observed;

b) considerable erosion requires fettling with crusts and alloy taken from the hearth;

c) when bath failure is imminent the power should be turned off and the banks should be fettled with alloy waste and alloy from the hearth.

5. Appearance of poor, silicon-bearing ferrotungsten is possible at slag tapping when the hearth has been frozen high. The slaghole level should be kept 100-150 mm above the hearth bottom to prevent the outflow of the alloy.

A triangular pyramid made of iron chips and coke is erected underneath the electrodes prior to applying power after a down-time. The rate at which the power input is raised depends on the duration of down-time. If the electrodes are cooled down to the temperature of ambient air, the full load may be attained in three hours.

The heating up of lining after a major overhaul is first effected with firewood, the flames being constantly kept long to bake the electrodes. Following the consumption of 20 m³ of firewood in the course of four shifts, foundry coke is charged and heating is continued for another four shifts. Further heating is effected by electric current. Prior to turning on the power 3,000 kg of ferrotungsten slag are charged into the bath and a triangular heap of iron chips and coke is made up underneath the electrodes to facilitate the power input rise.

Power is applied to the first voltage tap (178 V). Baking of electrodes and heating up of the lining are effected in accordance with the following time-table:

<i>First shift</i>	<i>Second shift</i>	<i>Third shift</i>
2,350 A 1 hr	Power off . . . 2 hrs	Power off . . . 2 hrs
Power off . . . 1 hr	5,290 A . . . 2 hrs	7,050 A . . . 2 hrs
2,350 A 1 hr	Power off . . . 2 hrs	Power off . . . 2 hrs
Power off . . . 1 hr	5,875 A . . . 2 hrs	
2,940 A 1 hr		Total: 6 hrs
Power off . . . 1 hr	Total: 8 hrs	
4,110 A 2 hrs		
Total: 8 hrs		

During the last two hours of the third shift power input is gradually increased to full load—8,110 A.

Throughout the lining heating-up period, concentrates and small amounts of coke are fed underneath the electrodes. Care is taken to prevent electric arcs from damaging the hearth bottom beneath the electrodes.

In the fourth shift the furnace is operated at full load, without interruptions, with the concentrates and coke being charged at a low rate to melt up the metallic ingot.

In the fifth shift and further on the furnace operates at full load without interruptions in smelting up the ingot.

Iron chips are added in the amount to produce a 75% W alloy.

Following the accumulation of a sufficient amount of metal (~15 tons) scooping is begun and the furnace sides are intensely fettled to form a metallic bowl. The heating up of the lining, accumulation of the metallic ingot and formation of the metallic protective layer take about ten days.

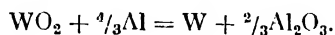
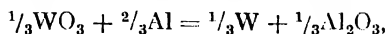
Scooped ferrotungsten is loaded into a dressing drum, where the ingot surface is cleared of slag. The drum is rotated for 40-45 minutes. After the drum is unloaded, the heat is sampled to obtain a complete chemical analysis. Standard production is packed onto wooden or metallic tare.

Specific raw-material and electric power consumption per basic ton of tungsten as well as tungsten recovery in the ferrotungsten production process are as follows:

Tungsten concentrate (60%), kg	1,529
Ferrosilicon 75%, kg	110
Iron chips, kg	70
Pitch coke, kg	115
Anode paste, kg	75
Steel rods, kg	55
Electric power, kWh	3,500
Tungsten recovery, %	99.0

Aluminothermic Production of Ferrotungsten

Ferrotungsten may also be produced without the use of furnaces, by means of tungsten oxide reduction by aluminium:



The evolving heat is sufficient to melt the mixture completely and to provide adequate metal separation from the slag. This method is now used in the U.S.S.R. to manufacture ferrotungsten with a high tungsten content.

In one of the Soviet smelters ferrotungsten is smelted from scheelite concentrates by the aluminothermic method in an electric furnace.

Ferrotungsten smelted by the aluminothermic method is up to specifications set in the U.S.S.R. (Table 51).

Table 51

Ferrotungsten Chemical Composition

Grade	Chemical composition, %								
	W minimum	Mn	Cu	S	P	C	Si	Al	Mo
		maximum, %							
B1	85	0.2	0.1	0.04	0.03	0.2	0.7	3.0	5.0
B2	89	0.2	0.1	0.04	0.03	0.2	0.5	3.0	6.0

The mixture composition for aluminothermic smelting is as follows, kg:

Scheelite concentrate	2,500
Primary aluminium grains	500
Iron chips	70
Iron ore	30-40
Lime	50

Lime is added separately following the melt-down of the mixture. Smelting is effected with bottom priming in an electric furnace hearth lined with magnesite brick. The mixture is fed to the furnace by a screw feeder. The duration of aluminothermic smelting is approximately 30 minutes; the electrodes are then lowered and the melt is heated by electric current for 30 minutes, after which slag is tapped. The metal and slag remnants are heated for an hour. Slag

with up to 0.5% tungsten trioxide is dumped, while richer slag is remelted. The bath is cooled for 16 hours, the metal ingot is recovered, cooled with water and dressed.

Consumption of materials per ton of ferrotungsten by this smelting method is as follows:

Ingot aluminium, kg	322
Scheelite, kg	1,550
Iron ore, kg	28
Electrodes 150 mm, kg	22
Electric power, kWh	1,059
Lime, kg	14
Iron chips, kg	37.6
Recovery of tungsten, %	97-98

Chapter 6

FERROTITANIUM

Designation and Composition of Ferrotitanium

Titanium is used in the manufacture of steel as a reducer and alloying element.

By its reducing capacity, titanium is considerably superior to silicon and manganese and inferior only to aluminium and zirconium, though it has a number of advantages over them. These advantages are that the products of titanium reduction have low melting points and are readily reverted to slag.

Titanium neutralises the nitrogen effect upon steel by bonding it into insoluble compounds.

Titanium is not used as an independent alloying element, but it is widely employed in combination with other elements. It combines with carbon in the manufacture of stainless and heat-resisting steels to improve their welding qualities and corrosion resistance. Steels treated with titanium or containing certain amounts of it usually possess higher mechanical properties, this being the result of efficient metal degassing and uniform distribution of sulphur inclusions.

Titanium is used in the form of ferrotitanium, whose composition, as specified by POCT 4761-54, is given in Table 52.

Table 52

Chemical Composition of Ferrotitanium

Grade	Chemical composition						
	Ti % minimum	C, %	$\frac{Si}{Ti}$ ratio	P, %	S, %	$\frac{Al}{Ti}$ ratio	Cu, %
		maximum					
Ti0	25.0	0.15	0.18	0.05	0.05	0.25	3.0
Ti1	23.0	0.15	0.20	0.05	0.05	0.27	3.0
Ti2	23.0	0.20	0.28	0.08	0.08	0.40	4.0

Raw Materials

Titanium occurs widely in nature and is a component of various minerals which contain its oxides. But only a few minerals are commercial: rutile, ilmenite, titanite (sphene) perovskite.

Rutile (TiO_2) is a reddish mineral with up to 90% TiO_2 in the ore.

Ilmenite ($\text{FeO} \cdot \text{TiO}_2$) is a black mineral with metallic lustre.

Both these minerals are found in the Urals.

The chief raw materials for the production of ferrotitanium are complex iron-titanium ores—the titanomagnetites which contain ilmenite in combination with magnetic iron ore. The titanium dioxide TiO_2 content in these ores varies from 4.5 to 16%.

Concentrates with a composition corresponding to specified standards are obtained by ore dressing (Table 53).

Table 53

Chemical Composition of Concentrates, %

Components	I grade	II grade	III grade
Iron oxide, maximum	53.6	53.6	53.6
Titanium dioxide, minimum	42	40	38
Silica, maximum	2.5	2.5	4
Moisture, maximum	7	7	10

Physicochemical Principles of the Smelting Process; Ferrotitanium Manufacturing Methods

Titanium is one of the most widely distributed elements in nature: its content in the earth's crust comes to 0.61%.

Pure titanium is a metal grey in colour with a silvery lustre, possessing the following physicochemical properties:

Atomic weight	47.90
Specific gravity	4.51
Melting point, °C	1660
Boiling point, °C	3260
Valence	2, 3 and 4
Latent heat of fusion, cal/mol	4,353

Titanium forms two compounds with iron.

Of these FeTi is a stable compound while Fe_2Ti exists only in cemented carbide tool alloy.

With carbon titanium forms a stable carbide TiC , its heat of formation being 57,000 cal. The melting point of TiC is 3100°C , its specific gravity 4.25.

With silicon titanium forms silicides Ti_3Si_3 , TiSi and TiSi_2 ; of these the most stable is Ti_5Si_3 ; its melting point is 2120°C .

Titanium forms the following main oxides with oxygen: titanium dioxide TiO_2 , titanium oxide Ti_2O_3 and titanium monoxide TiO . Lower oxides are basic in character; titanium dioxide is amphoteric. The properties of titanium oxides are shown in Table 54.

Table 54

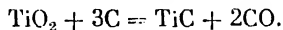
Properties of Titanium Oxides

Properties	Oxide		
	TiO_2	Ti_2O_3	TiO
Colour	White	Violet-black	Copper-red
Specific gravity	4.20	4.60	4.93
Melting point, $^\circ\text{C}$	1850	2130	2020
Heat of formation, cal/mol	219,000	375,500	135,090

The high values of the heats of formation of titanium oxides are signs of their stability.

Titanium dioxide may be reduced by carbon in an electric furnace. The reduction of titanium dioxide proceeds by stages, with the result that lower oxides form first.

Since titanium forms a chemically stable compound with carbon—titanium carbide—the reduction of titanium oxides brings not metallic titanium, but titanium carbide



The resulting titanium carbide dissolves in the iron introduced into the mixture by iron chips or by ore; the alloy obtained is called ferroc carbon titanium. The composition of the alloys obtained is 15-20% Ti, 5-8% C, 1-3% Si; the remaining percentage is iron and other impurities.

Because of its high carbon content this alloy may be used for deoxidising or degassing carbonaceous steels; it is unfit as an alloying addition in the smelting of stainless and other special steels.

The carbon content in the alloy may be lowered by the introduction of silicon; this binds titanium into stable compounds, titanium silicides. The alloy bears the name of silicotitanium and contains 20-25% Ti, 20-25% Si and a maximum of 1% C. A silicon-aluminium-titanium alloy may be obtained when part or the whole of quartzite in the mixture is replaced by bauxite; the composition of this alloy

is 30% Ti, 12-14% Al, 8-10% Si. These alloys find limited use because of the high silicon and aluminium content.

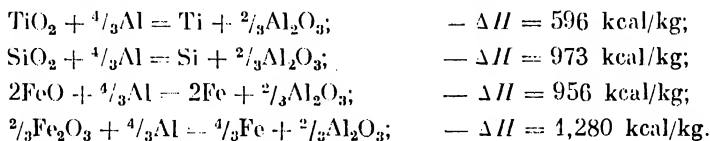
Titanium recovery in these processes averages 70-75%.

The foremost method of ferrotitanium manufacture is aluminothermic reduction, which proceeds as follows:



In the reduction of concentrates by aluminium the iron oxides are reduced almost completely (99%), silicon by 80%, titanium dioxide by 70-75%. The titanium remains in the slag in the form of lower oxides (TiO and Ti_2O_3), which are hard to reduce.

The chemistry of concentrate reduction by aluminium is as follows:



Titanium dissolves in iron in the process of ferrotitanium manufacture and forms compounds of titanium with aluminium. This facilitates the flow of reduction reactions towards the formation of titanium and increases the carry-over of titanium into the alloy.

Titanium monoxide, being a strong base, may form compounds with alumina and silica which act favourably upon the flow of reactions of reduction towards the formation of titanium monoxide. To counter the said process stronger basic oxides, capable of replacing TiO in its compounds with alumina, should be introduced into the mixture. CaO is usually used as such a base. That increases titanium reduction, but lowers the temperature of the process and, consequently, augments metal losses in the slag (in the form of trapped particles). As a result, the positive effect of lime is lowered drastically. The optimum amount of lime should average 20% of the aluminium weight.

An increase in the amount of aluminium in the mixture leads to a higher titanium carry-over to the alloy because this shifts the equilibrium of the reduction reaction of TiO_2 to Ti towards the formation of titanium.

But it also increases the aluminium content in the alloy.

Pre-heating of the mixture is of importance, for it allows iron ore additions to be decreased or even eliminated completely and, therefore, increases the titanium content in the alloy, decreases aluminium consumption and lowers the slag to alloy ratio as well as titanium losses.

The pre-heating of the mixture increases the specific heat of the process by approximately 30 kcal/kg for every 100°C .

The optimum mixture pre-heating temperature is 190-210°C. An excessive pre-heating temperature may lead to the oxidation of iron monoxide to oxide and thus may cause additional aluminium consumption and worsen operating results.

The slag in ferrotitanium smelting is composed approximately of 10-12% Ti, 0.5-1.2% SiO_2 , 8-11% CaO, 2-4% MgO, 0.7-1.2% FeO and 65-68% Al_2O_3 . Titanium in the slag is mainly in the form of lower oxides. The slag to alloy ratio is 1.3-1.4.

Smelting Technology

Ferrotitanium is smelted by the aluminothermic method in cast-iron sectional shafts up to 2 m in diameter and 1.6 m high; the walls of the shaft are approximately 60 mm thick. The shaft is either mounted on a special carriage equipped with cast-iron stools or installed on the shop floor. The slits in the joints are luted up with fireclay. The bottom is covered with a 200 to 250 mm layer of magnesite powder.

The lower part of the shaft (where the melted metal is collected) is lined with magnesite brick or with a 50 to 60 mm layer of castable magnesite refractory which is tamped in between the shaft's walls and a special pattern made of roofing iron. The magnesite powder used for the lining of the smelting shafts should be heated to 300-400°C. In some smelters tamped shafts are used for smelting ferrotitanium. In this case the shaft's casing is made of 5 to 7 mm steel plate while the walls are lined with tamped magnesite powder. The tamping is 200-300 mm thick.

A section of a cast-iron shaft is shown in Fig. 80.

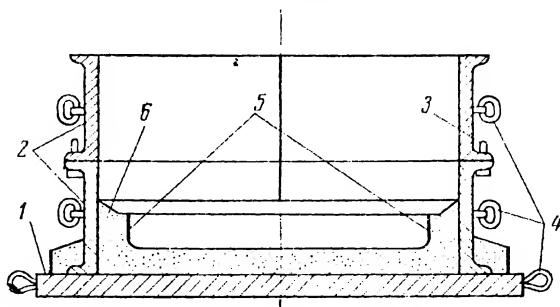


Fig. 80. Shaft for smelting ferrotitanium:

1—cast-iron stool; 2—sectional cast-iron shaft; 3—pins with wedges for setting up shaft; 4—rines for the transport of the shaft by an overhead crane; 5—pattern from roofing iron; 6—magnesite powder filling

The mixture for smelting ferrotitanium is composed of ilmenite concentrate, aluminium grains, ferrosilicon, iron ore and lime.

The concentrates are delivered to smelters in the form of powder with particles up to 1.0 mm in size. It is recommended to average concentrates, for concentrates from different deposits vary sharply in chemical composition. The concentrates to be smelted should have a sulphur content of not more than 0.05% and should be dried to zero moisture. Concentrates with a higher sulphur content are subjected to oxidising roasting at 950-1050°C which oxidises sulphur and eliminates it with furnace gases in the form of SO₂. Titanium concentrate is roasted in rotating kilns about 25 m long and 2.5 m in diameter; it is fired with coke and yields 120 tons of concentrates a day.

Calclne is screened through a 3-mm sieve to eliminate sintered lumps of concentrate, and is smelted while hot, thus making preheating of mixture unnecessary.

Aluminium grains are manufactured from secondary aluminium by pulverisation of liquid metal with the aid of compressed air; the grains should not exceed 3 mm and 85% of particles should be from 0.1 to 1.5 mm in size.

The main impurities in aluminium are silicon, copper, iron, magnesium, manganese and zinc. Silicon is a useful addition as it improves the reducibility of titanium from concentrates. This is explained by the fact that silicon, which partly replaces aluminium as a reducing agent, lowers the Al₂O₃ content in the slag, thus facilitating the reduction of titanium dioxide. Besides, together with titanium, silicon forms silicides, thus facilitating the titanium carry-over into the alloy.

Iron and copper revert to the alloy without taking part in the reactions; magnesium, manganese and zinc are harmful because they lower the reducibility of titanium and worsen the quality of the alloy.

The only iron ore used is the hematite which contains at least 96% Fe₂O₃; prior to smelting it is dried to zero moisture and sieved through a 3-mm sieve.

Lime should be freshly burnt (with the CaO content being minimum 90%) and crushed down to lumps of 3 mm in size (prior to its smelting crushed lime should not be stored for more than eight hours; the use of hydrated and slaked lime is inadmissible).

Ferrosilicon, used as a silicon-bearing addition when working with low-silicon aluminium, should be free of foreign impurities and dried, and the size of its grains should not exceed 1 mm. The composition of the mixture is determined by charge calculations. A simplified charge calculation for ferrotitanium smelting is given

below. This estimate is similar to those used in the metallothermic production of other ferroalloys.

Charge Calculations

The calculation is made for 100 kg of ilmenite concentrate. The assay of the mixture materials used for smelting ferrotitanium is given in Table 55.

Table 55

Composition of Mixture Materials

Materials	Composition, %			
	TiO ₂	Fe _{total}	SiO ₂	Al
Ilmenite concentrate	42.0	36.0	3.0	—
	Cu		Si	
Aluminium powder	4.0	1.0	1.5	90.0
	CaO			
Lime	90%	—	—	—
	Fe ₂ O ₃			
Iron ore	98%	—	—	—

In determining the amounts of FeO and Fe₂O₃ present in the concentrate it is assumed that the quantity of FeO in the concentrate corresponds to the formula of ilmenite FeO·TiO₂, while all the remaining iron is in the form of Fe₂O₃.

Two kg of ore should be used per 100 kg of concentrates in smelting a pre-heated mixture and 11 kg in smelting a cold mixture.

The amount of lime in the mixture averages 9% of the concentrate weight.

Proceeding from smelter practice, we assume that:

1. 75% of TiO₂ is reduced to Ti; 12.5% TiO₂ to TiO (slag) and 12.5% to Ti₂O₃ (slag).

2. 99% of Fe₂O₃ is reduced to Fe and 1% to FeO; 99% of FeO in the concentrate is reduced to Fe.

3. 90% of SiO₂ is reduced to Si.

4. Al to Ti ratio in the alloy equals 0.25.

5. Secondary aluminium powder is used as a reducer.

Ferrosilicon should be added to the mixture when using low-silicon aluminium; this improves the reduction of titanium. Addition of excessive amounts of ferrosilicon leads to a rise in the silicon content in the alloy, i.e., to production of grade Tn2 ferrotitanium.

6. The amount of reducer in the mixture is assumed to be 110% of the theoretical amount necessary for the reduction of titanium, iron and silicon oxides and for the carry-over of aluminium into

the alloy. The excess of 10% of aluminium is explained by the fact that aluminium is used partly for the reduction of MnO , V_2O_5 and other uncalculated oxides contained in the mixture, and by the fact that lower oxides of iron partly oxidise to Fe_3O_4 in the roasting process.

7. With the exception of Al_2O_3 impurities in aluminium powder revert to the slag entirely.

Let us determine the FeO and Fe_2O_3 assay in the concentrate. In accordance with the formula of ilmenite $\text{FeO} \cdot \text{TiO}_2$ the concentrate contains:

$$42.0 \times \frac{72}{80} = 37.8 \text{ kg of FeO, or 40\%;}$$

iron contained in the 37.8 kg of FeO :

$$37.8 \times \frac{56}{72} = 29.4 \text{ kg;}$$

contained in the form of Fe_2O_3 :

$$36 - 29.4 = 6.6 \text{ kg;}$$

amount of Fe_2O_3 in the concentrate:

$$6.6 \times \frac{160}{112} = 9.44 \text{ kg.}$$

Oxides reduced from concentrates and ores:

TiO_2 to Ti

$$42.0 \times 0.75 = 31.5 \text{ kg;}$$

TiO_2 to Ti_2O_3

$$42.0 \times 0.125 = 5.25 \text{ kg;}$$

TiO_2 to TiO

$$42.0 \times 0.125 = 5.25 \text{ kg;}$$

FeO to Fe

$$37.8 \times 0.99 = 37.4 \text{ kg;}$$

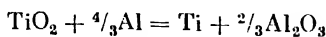
Fe_2O_3 to Fe

$$(2.00 \times 0.98 + 6.6) + 0.99 = 8.5 \text{ kg;}$$

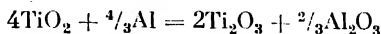
SiO_2 to Si

$$3.3 \times 0.90 = 2.7 \text{ kg.}$$

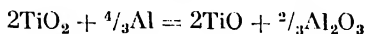
Aluminium required for the reduction of oxides:



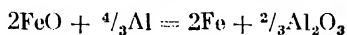
$$31.5 \times \frac{36}{80} = 14.35 \text{ kg};$$



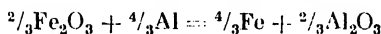
$$5.25 \times \frac{36}{320} = 0.59 \text{ kg};$$



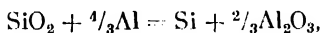
$$5.25 \times \frac{36}{160} = 1.18 \text{ kg};$$



$$37.4 \times \frac{36}{144} = 9.35 \text{ kg};$$



$$8.5 \times \frac{36}{106.6} = 2.96 \text{ kg}.$$



$$2.7 \times \frac{36}{60} = 1.64 \text{ kg}.$$

Total amount of aluminium required for the reduction of oxides:

$$14.35 + 0.59 + 1.18 + 9.35 + 2.96 + 1.64 = 30.07 \text{ kg}.$$

Aluminium reverted to the alloy:

$$31.5 \times \frac{48}{80} \times 0.25 = 4.73 \text{ kg}.$$

Total amount of aluminium necessary, including the 10% excess:

$$(30 + 4.73) \times 1.10 = 38.28 \text{ kg}.$$

Amount of aluminium powder necessary:

$$\frac{38.28}{0.9} = 42.5 \text{ kg}.$$

Composition of the mixture, kg:

Ilmenite concentrate	100.0
Iron ore	2.0
Aluminium powder	42.5
Lime	9.0
	<hr/>
	153.5

8. Weight and composition of the alloy:

	kg	%
Ti	$31.5 \times \frac{48}{80} = 18.9$	29.1
Fe	$36.0 \times 0.99 + 42.5 \times 0.01 = 36.03$	55.5
Al	4.73	7.27
Si	$2.7 \times \frac{28}{60} + 42.5 \times 0.015 = 1.90$	2.93
Cu	$42.5 \times 0.04 = 1.70$	2.62
Other impurities (2.5%)	1.74	2.58
Total amount of alloy produced: 65.00		100.00

Heat evolved in oxide reduction:

$\text{TiO}_2 + \frac{4}{3}\text{Al} = \text{Ti}$	$+ \frac{2}{3}\text{Al}_2\text{O}_3;$	$\Delta H = 47,600 \text{ cal};$
$4\text{TiO}_2 + \frac{4}{3}\text{Al} = 2\text{Ti}_2\text{O}_3$	$+ \frac{2}{3}\text{Al}_2\text{O}_3;$	$\Delta H = 141,800 \text{ cal};$
$2\text{TiO}_2 + \frac{4}{3}\text{Al} = 2\text{TiO}$	$+ \frac{2}{3}\text{Al}_2\text{O}_3;$	$\Delta H = 98,600 \text{ cal};$
$2\text{FeO} + \frac{4}{3}\text{Al} = 2\text{Fe}$	$+ \frac{2}{3}\text{Al}_2\text{O}_3;$	$\Delta H = 137,670 \text{ cal};$
$\frac{2}{3}\text{Fe}_2\text{O}_3 + \frac{4}{3}\text{Al} = \frac{4}{3}\text{Fe}$	$+ \frac{2}{3}\text{Al}_2\text{O}_3;$	$\Delta H = 136,500 \text{ cal};$
$\text{SiO}_2 + \frac{4}{3}\text{Al} = \text{Si}$	$+ \frac{2}{3}\text{Al}_2\text{O}_3;$	$\Delta H = 58,367 \text{ cal}.$

Heat evolved in the reduction of 100 kg of ilmenite concentrate, kcal:

TiO_2 to Ti	$31.5 \times \frac{47,600}{80} = 18,774;$
TiO_2 to Ti_2O_3	$5.25 \times \frac{141,800}{320} = 2,326;$
TiO_2 to TiO	$5.25 \times \frac{98,600}{160} = 3,239;$
FeO to Fe	$37.4 \times \frac{137,670}{144} = 35,754;$
Fe_2O_3 to Fe	$8.5 \times \frac{136,500}{107} = 10,880;$
SiO_2 to Si	$2.7 \times \frac{58,367}{60} = 2,627;$
Uncalculated oxides (10% of total)	7,360

Total heat evolved 80,960 kcal.

Specific heat of the process:
from reduction processes

$$\frac{80,960}{153.5} = 527 \text{ kcal/kg};$$

from pre-heating of the mixture (to 200°C)

$$30 \times \frac{200}{100} = 60 \text{ kcal/kg};$$

Total: $527 + 60 = 587 \text{ kcal/kg}.$

which exceeds the 550 kcal per kg necessary for the metallothermic process.

The weight of materials per heat depends on the size of the shaft and the smelting methods, and usually ranges from 1,500 to 5,000 kg of concentrates and corresponding amounts of other materials.

Mixture materials are accurately weighed and then thoroughly mixed. To facilitate the mixing of certain components, the materials are weighed in the following order: aluminium powder, lime, ferro-silicon and then concentrates and iron ore.

In the proportioning process the temperature of the ilmenite concentrate should not exceed 600°C, since higher temperature may cause spontaneous ignition of the mixture.

The mixture is mixed in drum-type mixers or by repeated chargings into special conical bins.

Ferrotitanium is smelted with bottom priming which ensures a more uniform flow of the process; from 200 to 400 kg of prepared mixture are charged onto the shaft bottom and ignited by means of a priming mixture composed of sodium nitrate, aluminium powder and electron turnings. The priming mixture is ignited by a red-hot iron rod or an electric spark.

After the beginning of the aluminothermic process the remainder of the mixture is charged from a special hopper equipped with a gate and a charging pipe; care is taken to ensure that the charge is uniformly spread over the whole shaft area and that the mixture is fed at a rate so that the melt surface is constantly covered with a thin layer of it. The slow charging rate causes the uncovering of the melt surface and, consequently, increases heat losses. A too rapid charging of the mixture may lead to a high content of aluminium in the alloy.

The normal run of a heat is characterised by rapid and uniform consummation of reactions over the entire melt surface and by uniform evolution of gases during the heat. The heat lasts from 10 to 15 minutes, depending on the weight of the concentrate.

With lack of reducer, low specific heat of the process or insufficient pre-heating of the mixture, the heat is cold, the reactions go slowly and the consummation of the heat is delayed.

The violent run of the heat with irregular evolution of gases and eruptions of mixture are a sign of moisture in the mixture materials or in the magnesite filling. Poor mixing of materials may be the cause of the anomalous run of the heat.

These deviations in the run of the heat may cause spoilage and lower performance results.

Following the consummation of the heat, after a delay of 10-15 minutes which is necessary to settle the metal prills, the slag is

drained in such a manner that the metal ingot remains covered with a layer of slag at least 100 mm thick. To make better use of smelting shaft volume, and to economise on refractories an intermediate tapping of the slag is sometimes effected, following the charging of $\frac{3}{4}$ of the whole amount of the mixture. The slag is tapped through a taphole in the side of the shaft after a 10-minute killing.

An iron thermit mixture (settler) is sometimes added to the melt to facilitate the settling of prills in the slag. The mixture is composed of the following (at one of the smelters), kg:

Iron ore	420
Aluminium powder	105
75% ferrosilicon	110
Lime	60

Following the heat, the thermit mixture is uniformly spread over the surface of the liquid slag. This keeps the slag overheated, thus promoting the settlement of metal inclusions, while the reduced iron from the iron-thermit mixture dissolves titanium and enlarges the metal particles as a result of which the recovery of titanium increases to 77%.

Electric heating of slag in the aluminothermic smelting also helps settle the prills in the furnace crucible. After the consummation of the heat, power is applied to the furnace and the settler (iron ore, crushed ferrosilicon and lime) is added by small portions onto the melt surface. Following the melt-down of the settler the melt is held under current for 15-20 minutes. This method secures a sharp decrease in aluminium consumption (approximately down to 400 kg per ton of alloy), but the recovery of titanium remains at 70-72%.

The Lipetsk Ferroalloy Works has developed a method of ferro-titanium manufacture in a tilting closed bath with separate tapping of slag and metal. The slags are processed by a settler composed of iron ore, aluminium powder and lime to improve their fluidity and to make prills settle more completely.

The heats are conducted with bottom priming. The settler is added after the heat, the melt is delayed for five-eight minutes, and the bulk of the slag is then drained.

The rest of the slag is processed with lime, heated to 600-800°C; lime is mixed with the melt and then tapped simultaneously with the metal. This slag is free from prills and separates itself readily from the metal. This method sharply increases productivity and saves on refractories and aluminium, especially when the slag is electrically heated.

Experimental heats, run in slagged steel ladles, have been conducted at the Aktyubinsk Ferroalloy Works in accordance with the following technological flowsheet.

The mixture is added onto the slag left over in a ladle from the preceding heat, and the usual melting procedure is followed. The slag crusts formed at the beginning on the ladle walls protect them from corrosion. After the consummation of the heat the slag is poured into another ladle, while the metal, with a thin skin of slag protecting it against volatilisation, is teemed into moulds.

The metal ingot obtained in the shaft smelting is held over for eight hours inside the shaft, then cooled in water, smashed into lumps weighing less than 10 kg for grades Tn0 and Tn1 and maximum 15 kg for grade Tn2, and after that packed into wooden cases. Ferrotitanium is shipped to consumers by heat batches; the amount of fines passed through a 2.5-mm sieve should not exceed 8%.

The chemical composition of commercial ferrotitanium heats is given in Table 56.

Table 56

Chemical Composition of Commercial Ferrotitanium Heats

Ti	Si	Al	C	S	P
29.01	3.48	5.62	0.10	0.02	0.03
30.52	3.32	7.18	0.08	0.01	0.03
25.42	4.60	5.95	0.05	0.02	0.02

Recovery of titanium in ferrotitanium manufacture equals 72%.

The consumption of raw materials per basic ton of alloy (20% Ti) is as follows:

Ilmenite concentrate (38% TiO ₂)	1,200
Secondary ingot aluminium	483
75% ferrosilicon	10.5
Iron ore	90.0
Lime	100.0
45% ferrosilicon	7.1

The distribution of ferrotitanium according to grades (in % of the total amount manufactured):

Tn0—50%, Tn1—35%, Tn2—15%.

Chapter 7

PRODUCTION OF FERROMOLYBDENUM

Designation and Composition of Ferromolybdenum

Ferromolybdenum is an alloying addition which sharply raises the mechanical and physical properties of steels.

The addition of molybdenum ensures uniform microcrystalline structure in steel, augments its hardenability and eliminates post-temper brittleness. Structural steels with 0.15-0.50% Mo possess greater impact strength, and that is especially valuable for the manufacture of such details as automobile shafts, gears, rolls, etc.

Molybdenum is used for the manufacture of high-speed steels, which are used for various cutting instruments—drills, milling cutters, cutting tools, etc.—capable of withstanding high-cutting speeds.

Molybdenum is widely employed in the manufacture of stainless, heat-resisting and acid-resisting steels.

The chemical composition of ferromolybdenum is specified by ГОСТ 4759-49 (Table 57).

Table 57

Chemical Composition of Ferromolybdenum

Grade	Chemical composition, %							
	Mo mini- mum	C	Si	S	P	Sb	Cu	Sn
		maximum						
Mo1	55	0.10	1.00	0.10	0.10	0.05	0.80	0.05
Mo2	55	0.15	1.50	0.15	0.15	0.08	1.50	0.08
Mo3	55	0.20	2.00	0.20	0.20	0.10	2.50	0.10

Ferromolybdenum is delivered in lumps maximum 5 kg in weight; the admissible fines content (sieved through a 10-mm sieve) is 10%. Ferromolybdenum is packed in wooden cases or steel drums.

Raw Materials for Smelting Ferromolybdenum, Their Delivery Specifications and Pre-Smelting Preparation

Ores. Molybdenum is the 37th element in occurrence among the elements making up the earth's crust; its content in the latter comes up to 0.0003%.

Pure molybdenum is not found in nature. The chief ore minerals of molybdenum are molybdenite and wolfenite. Molybdenite—molybdenum sulphide MoS_2 —is outwardly like graphite, but is twice as heavy (sp. gravity 4.7). The pure mineral contains 59.95% Mo and 40.05% S. Wolfenite is a lead molybdate PbMoO_4 (sp. gravity 6.7-7.0), yellow-orange in colour. The pure mineral contains 25.15% Mo.

Molybdenum minerals are dissipated and their content in the ores is negligible. Ores containing 0.5% Mo are considered to be commercial; however, prior to metallurgical smelting they should be dressed to obtain concentrates rich in molybdenum. Their composition, set by GOCT 212-48, is given in Table 58.

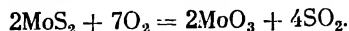
Table 58

Chemical Composition of Molybdenum Concentrates

Grade	Impurities content, %						Approximate use
	Mo mini- mum	SiO ₂	As	Sn	P	Cu	
		maximum					
KM1	50	5	0.07	0.07	0.07	0.5	Manufacture of grade Mo1 ferromolybdenum
KM2	48	7	0.07	0.07	0.07	1.0	Manufacture of grades Mo2 and Mo3 ferromolyb- denum
KM3	47	9	0.07	0.07	0.15	2.0	

A raw molybdenum concentrate containing 37-40% S is subjected to oxidising roasting in multi-hearth roasters to eliminate sulphur. A multi-hearth roaster, shown in Fig. 81, is composed of iron cylindrical shell 1, lined internally with firebrick. Central shaft 2, situated in the centre of the roaster, is driven by an electric motor through a system of gears 3. Rabbles 4 with blades 5 are mounted on the central shaft. The raw concentrate is charged onto the first (upper) hearth of the roaster, then is moved by blades and then falls through openings in the hearths. At the end of the roasting process it is discharged through discharge opening 6. A roaster at a 5-7 hearth level is shown in Fig. 82.

Reaction of molybdenum concentrate roasting



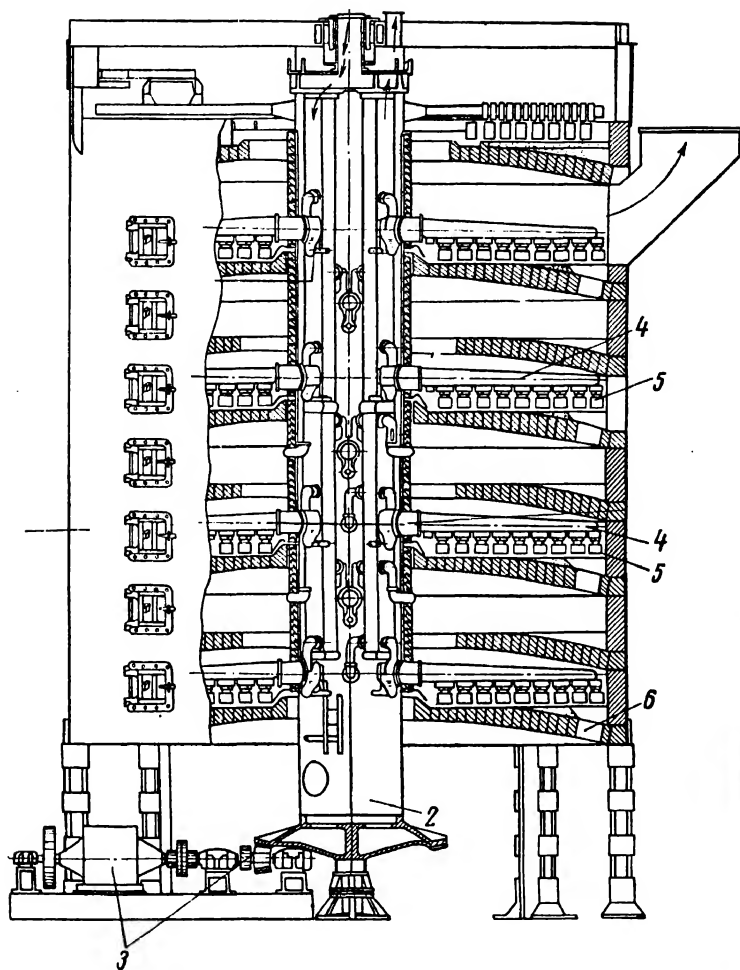


Fig. 81. Multi-hearth roaster

Oxidation of molybdenum sulphide with the formation of molybdenum oxides begins at 360°C for a concentrate with particles less than 0.063 mm; with the increase in the size of particles to 0.09-0.127 mm the temperature of spontaneous ignition rises approximately to 500°C.

The speed of the reaction grows with the increase in temperature. However, MoO_3 possesses high volatility which increases with the temperature; therefore, the maximum permissible roasting temperature should not exceed 650-700°C.

The oxidising reaction is attended by a considerable evolution of heat, and additional heating by gas is necessary only in the final stage of the roasting, when sulphide content is negligible.

Normal-run roasting conditions require adequate mixing of the material roasted, oxygen supply and removal of sulphurous gases from the reaction zone by means of a forced draft. Of major importance for the normal run of the process are uniform charging rate of the concentrate, constant molybdenum or sulphur content in the mixture and constant concentrate sizing (oxidation of molybdenite in

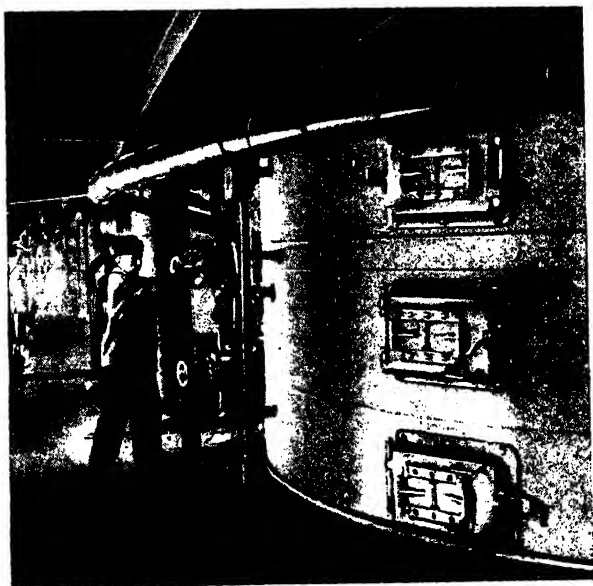


Fig. 82. Outer view of a multi-hearth roaster at 5th-7th hearth level

big lumps, with all other conditions remaining the same, is slower than in the case of dispersed particles with a greater reactive surface). If these conditions are observed the furnace yield amounts to 110-120 kg of roasted concentrate per square metre of hearth area per 24 hours.

The furnace is charged with 600-840 kg of concentrate per hour, depending on its composition:

Concentrate without reverts	600-620 kg/hr
Concentrate with reverts from electric precipitators and flues (15-20%)	690-710 kg/hr
Concentrate with addition of concentrate discarded because of sulphur, and reverts from roaster hearth (15-20%)	720-740 kg/hr
Ditto (20-30%)	820-840 kg/hr

Roasting operating conditions as to temperature should be as follows:

Hearth number	2	3, 4, 5	6, 7	8
Temperature, °C	400	Up to 700	600-650	Up to 500

The roasted concentrate is discharged continuously from the eighth hearth into a hopper (conveyor).

A roasted molybdenum concentrate assays approximately 80-90% MoO_3 , 3-8% FeO , 3-10% SiO_2 , 0.4-2.0% CuO , 0.05-0.15% S and 0.02-0.15% P.

The roasted concentrate is divided into three grades, depending on its sulphur content: grade 1 with up to 0.09% S, grade 2 with up to 0.10-0.12% S and grade 3 with up to 0.13-0.15% S.

If the sulphur content exceeds 0.15% the concentrate is discarded and reroasted.

One of the basic factors determining efficient multi-hearth roasting is the roaster campaign.

The roaster operation causes considerable wear of blades; accretions of concentrate formed on the hearth may force furnace shutdown.

Prior to being smelted the roasted concentrate is crushed in roll or hammer mills.

Reducers. A 75% ferrosilicon, with 76-77% Si for the first group and 77-78% Si for the second, is used to reduce the molybdenum and iron oxides introduced with the concentrate and iron ore.

Ferrosilicon is crushed into 10 to 20 mm lumps and then milled to powder in a ball mill. Powder with grains over 3 mm is not smelted. Moreover, aluminium powder manufactured from secondary aluminium and duralumin is used as a reducer to bolster heat evolution in the process.

Aluminium powder sizing should meet the following requirements (as per technical specifications):

Size 1 mm	Up to 5%
" 0.5-1.0 mm . . .	" " 25%
" 0.3-0.5 mm . . .	Minimum 20%
" 0.3 mm	Up to 45%
Flour (-190 mesh)	" " 15%

Iron-bearing materials. Iron ore is used as an iron-bearing material and additional source of oxygen necessary for optimum mixture heat capacity. At the same time, iron ore is a fluxing material because the iron oxide reverts partly to the slag and renders it more fluid, this being a necessary condition for the normal flow of the ferromolybdenum smelting process.

Iron ore should contain: Fe minimum 65%, P and S not over 0.05% each. Iron in the ore should be in the form of Fe_2O_3 .

Prior to being used the ore is dried in a drum-type drier (Fig. 83).

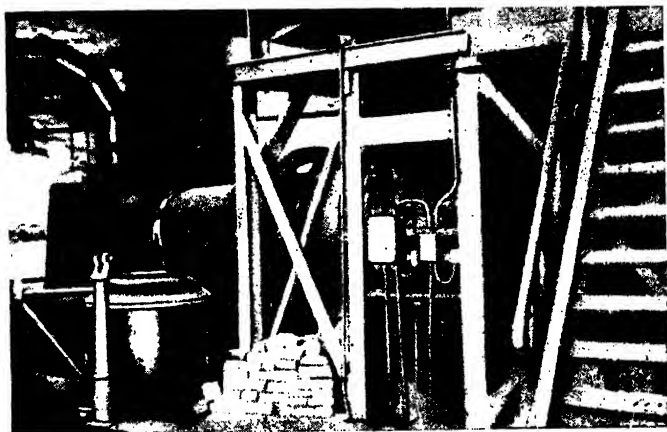


Fig. 83. Rotating kiln for ore drying

Dried ore with particles larger than 3 mm is milled in a ball mill.

Iron is partly introduced in the form of turnings (turnings of simple carbonaceous steels should contain no alloying additions, other than molybdenum). The turnings should be milled and the moisture and oil they contain should be eliminated.

Fluxing materials. Fluxing materials—lime and fluorspar—are used for slag fluidity. The CaO content in the lime should not be below 90%, while fluorspar should not contain less than 90% CaF_2 and not more than 5% SiO_2 . Both lime and fluorspar should be crushed to particles 3 mm and less in size.

Physicochemical Principles of Ferromolybdenum Production Process

Molybdenum is a silvery white metal with the following physicochemical properties:

Atomic weight	96
Specific gravity	10.2
Melting point, °C	2622
Boiling point, °C	4800
Latent heat of fusion, cal/g-atom	6,600
Specific heat capacity at 25°C, cal/g°C	0.065
Valence	1 to 6

Combined with carbon molybdenum forms two carbides— Mo_2C and MoC . With silicon molybdenum forms silicides MoSi_2 , Mo_2Si_3 and MoSi . With oxygen molybdenum forms oxides MoO_2 , Mo_2O_5 , MoO_3 and unstable oxide MoO .

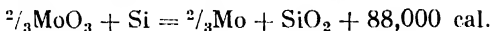
Molybdenum trioxide MoO_3 is met with in the smelting processes; it is a white powder with a greenish tint, which turns yellow when heated. Its specific weight is 4.69, its melting point 795°C . From 650°C upwards MoO_3 volatilises readily. With sulphur molybdenum forms sulphides Mo_2S_3 , Mo_2S_5 and MoS_2 .

With iron molybdenum forms alloys in any proportions. It is capable of giving solid-state compounds with iron— FeMo and Fe_7Mo_6 . Alloys with more than 50% of molybdenum possess high melting points (for example, an alloy with 60% Mo melts at 1800°C) which makes tapping of liquid metal quite difficult.

Molybdenum oxides are chemically unstable and may be readily reduced by carbon, silicon and aluminium (Fig. 84).

Molybdenum reduction by carbon: $\text{MoO}_3 + 3\text{C} = \text{Mo} + 3\text{CO}$, may be easily effected in an electric furnace. But it involves substantial losses of molybdenum, considerable electric power consumption and a number of difficulties in refining the alloy from carbon. Because of this, ferromolybdenum is now produced by the silicothermic method. This method is distinguished by the fact that silicon (added to the mixture in the form of ferrosilicon) is used as a reducer instead of carbon.

The reduction of molybdenum proceeds as follows:



Here the specific heat of the process equals 710 cal/kg (the process is practically feasible with the evolution of 500 cal of heat per kg of mixture).

The equilibrium of this reaction is strongly shifted to the right and it has been practically established that 99% of molybdenum oxides are reduced in the smelting process.

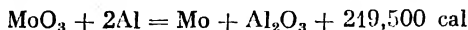
Iron is introduced into the mixture partly in the form of iron turn-

ings and partly in the form of iron ore. The oxidation of silicon by the oxides of iron ore is accompanied by an additional evolution of heat. Iron oxide in the ore is reduced by 48% to iron and by 52% to iron monoxide, while iron monoxide is reverted to the slag and promotes its fluidity.

The silicon of gangue and the products of molybdenum trioxide and iron oxide reduction form a highly siliceous slag which is remarkable for its high viscosity.

Slag composed almost entirely of silica is highly viscous despite the fact that it is heated considerably over its melting point.

Metallothermic processes result in the simultaneous formation (from the mixture) of liquid metal and slag which separate from each other as the result of the settling of drops of metal contained in the slag. As these processes last a very short time and the temperature of the alloy falls rapidly, it is of utmost importance to obtain fluid slags in order to prevent metal prills (droplets) from being trapped in the slag. This is achieved by introducing iron monoxide (contained in the iron ore, as mentioned above) and alumina into the slag. The latter is introduced into the slag by partly replacing the reducer—silicon—by aluminium. The reaction



considerably increases the amount of heat, which raises the temperature and, consequently, lowers the slag's viscosity. Besides, partial replacement of silicon by alumina in the slag reduces the latter's viscosity too.

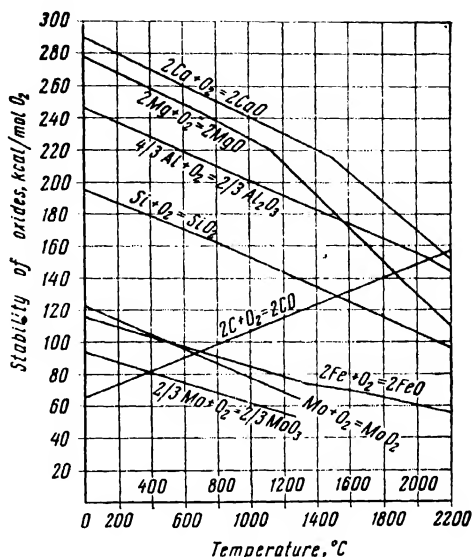


Fig. 84. Relationship between the stability of oxides, present in the raw materials used for the smelting of ferromolybdenum, and temperature

Technology of Smelting

Composition of the mixture. The mixture is calculated for 100 kg of concentrate. On the basis of smelting practice it is assumed that:

- 1) the amount of iron ore equals 26 kg per 100 kg of concentrate;
- 2) the amount of fluorspar and lime, depending on the nature of the concentrate, ranges from 4 to 6 kg. When working on concentrates with a high content of alkaline metals, addition of fluxes may be omitted;
- 3) Fe_2O_3 in the iron ore is reduced by 48% to Fe and by 52% to FeO. It is assumed that FeO in the concentrate is entirely reverted to the slag;
- 4) the aluminium to silicon ratio in the mixture should be about 1 to 10; the amount of aluminium powder per 100 kg of concentrate varies within the limits of 3.3-4.3 kg, depending on the molybdenum content in the concentrate. The maximum is taken when the concentrate contains less than 54.5% of molybdenum, the minimum when the molybdenum content is above 56%;
- 5) MoO_3 contained in the concentrate is reduced by 100% to Mo;
- 6) the excess of reducer (75% ferrosilicon) should not exceed 5% of the stoichiometric amount.

Approximate composition of a charge is as follows, kg:

Molybdenum concentrate	100.0
75% ferrosilicon	35.1
Iron ore	26.0
Iron turnings	19.5
Fluorspar	2.0
Lime	3.0
Aluminium powder	3.7

Nine charge portions make up one mixture batch which is thoroughly mixed in a mixing drum. Up to 36 charges are required for one heat.

Smelting procedure. Ferromolybdenum is smelted in a lined cylinder—a smelting shaft (Fig. 85) mounted on a sand pedestal with recesses ("nests") for the liquid alloy. The cylinder is lined with grade M-59 firebrick laid on edge. The "nest" is made of dry sand, and the thickness of a layer of fresh sand should not exceed 50 mm for each heat. Prior to installing the cylinder shaft on the "nest" the crust of the preceding heat should be removed from the lining, the taphole cleaned and the damaged lining patched up.

The taphole is plugged with a viscous lute made of refractory clay and sand and a wedge brick is run into it; the clearance between the taphole sides and the key stone is filled with sand and tamped.

Ferromolybdenum is smelted with surface priming which decreases molybdenum losses. The prepared mixture is charged evenly all

over the shaft bottom in order to prevent its fractioning. The level of the mixture is approximately 300 mm below the upper edge of the cylinder. Two or three holes are made on the surface for the priming mixture which is composed as follows, kg:

Aluminium powder	45
Saltpetre	5
Iron ore	15
Electron turnings	0.2

The priming mixture is ignited with a red-hot iron rod.

Prior to smelting, an evacuating hood is installed over the shaft and the exhaust ventilation is turned on. Normally the heat takes about 45 minutes. A plentiful concentrated evolution of gases is observed at the furnace-top when the heat runs normally.

When the furnace runs cold (feeble and irregular evolution of gases), an exothermic mixture with an excess of reducer should be added 15 to 20 minutes after priming.

The mixture may be composed as follows, kg:

Electron turnings	0.5
Aluminium powder	2.3
Iron ore	2.0

The total consumption of the exothermic mixture should not exceed 15-20 kg.

When the furnace runs very hot, discarded alloy may be added in the course of the heat; in normal conditions resmelting of discarded



Fig. 85. Smelting shaft. A forehearth for catching alloy is seen on the launder

alloy is permitted towards the end of the heat. The total amount of discarded alloy planned for resmelting should be calculated so as not to raise the silicon content in the alloy produced by more than 0.3%.

During the heat it is necessary to watch over the furnace lining. When hot-spots appear in the lower part of the shell they should immediately be covered with moist sand.

The heat is held over for 40-50 minutes following its conclusion (to settle metal prills) so as to prevent molybdenum losses in the form of metal prills in the slag. Following this delay the slag is tapped through a forehearth which is meant to catch the alloy particles. The slag is dumped if it contains no metal prills and its molybdenum assay does not exceed 0.3%. The shaft is then dismantled and the metal ingot cooled in the nest for 6-7 hours. Slag granulation with subsequent magnetic separation is now practised commonly.

Table 59 gives the composition of a number of commercial heats of ferromolybdenum.

The chemical composition of the slags of commercial ferromolybdenum heats is given in Table 60.

Table 59

**Chemical Composition
of Commercial
Ferromolybdenum Heats, %**

Mo	Si	P	S	C	Cu
61.16	0.16	0.033	0.08	0.04	0.47
64.67	0.80	0.046	0.08	0.10	0.48
59.51	0.15	0.036	0.08	0.05	0.52
60.40	0.93	0.04	0.07	0.05	0.54

Table 60

**Chemical Composition of Slags
of Commercial Heats, %**

Mo	SiO ₂	FeO	Al ₂ O ₃	CaO	MgO
0.06	69.36	6.98	12.69	7.59	1.37
0.14	66.56	10.58	10.96	6.00	3.31
0.12	68.72	9.79	12.41	7.22	1.25
0.17	69.90	10.71	10.03	5.59	0.91

For final cooling the metal ingot is placed in a tank which is then filled with water. There the alloy cools down completely for six hours; after that it is crushed into lumps of not more than 5 kg, cleaned and packed into steel drums.

Production Control Methods

No high-speed analysis of ferromolybdenum is made but the alloy is preliminary graded visually.

The standard alloy has a fine crystalline structure and dull fracture. The presence of lustrous "stars" in the fracture indicate

increased sulphur content. Lustrous fracture ("specular play") is a sign of a high silicon content in the alloy.

The slag of a normal technological process is characterised by the following symptoms:

- 1) the slag forms threads at tapping and sampling;
- 2) when cooled, the slag becomes vitreous and light blue to dark in colour;
- 3) the slag becomes moderately convex on solidification in the slag pot.

The presence of liquid slag (which does not form threads) and its dark colour on solidification are signs of an abnormal technological process and an elevated iron oxide content. In most cases the presence of liquid slag characterises increased molybdenum and silicon content as well as alloy contamination by non-metallic inclusions.

The presence in the slag of considerable amounts of metallic pills is indicative of high slag viscosity and requires correction of its composition.

In all abnormal cases it is necessary to check the factors which determine correct technology and correct mixing of materials before taking measures for subsequent heats.

Performance Results

One of the most important factors of production efficiency is the recovery of molybdenum with an average of 98.0%.

In the process of ferromolybdenum manufacture molybdenum-bearing reverts are available from the following sources:

- 1) roaster dust which is recovered in electrostatic precipitators;
- 2) dust which is formed in proportioning and mixing operations;
- 3) burden mixture and slag ejected from the shaft during the smelting process;
- 4) alloy dressing wastes;
- 5) brick of the shaft lining and "nest" sand containing semi-melted metal, rich slag and unsmelted mixture;
- 6) slag containing over 0.30% Mo.

Careful collection and full utilisation of the above-mentioned waste materials are of major importance for production. Dust waste is resmelted by the metallothermic process. The remaining reverts are resmelted in an open three-phase electric furnace with graphitised electrodes at 210 V and 4,100 A.

The furnace hearth, manufactured of 8-mm steel plate, is sectional and composed of two parts—the bottom and the cylinder. The hearth is lined with magnesite brick. The shell, after being assembled and lined with brick, is mounted on a carriage and wheeled in under the electrodes.

The process of ferromolybdenum manufacture from reverts in an electric furnace is similar to the puddling process with periodic slag-off.

Three to four mixture batches are smelted in eight hours; each batch comprises, kg:

Rich slag and hearth material	350-450
Reverts from alloy dressing	50-100
Molybdenum concentrate roasting waste (S up to 2.0%) . . .	15-30
Lime	80-100
Magnesite brickbat	20-30

After each slagging-off the hearth sides are fettled with magnesite powder.

The smelting of a ferromolybdenum ingot weighing approximately five tons takes four to five days, following which the hearth is wheeled out from under the electrodes, and the ingot is dressed. The alloy obtained contains usually 43-50% Mo with an impurities content within the standard specifications; production of a standard alloy as to molybdenum is also possible.

Raw-material and electric power consumption per basic ton of ferromolybdenum amounts to (resmelting of reverts included), kg:

Molybdenum concentrate (51% Mo)	1,203
Amount of molybdenum therein	638.5
Iron ore	290
Iron turnings	205
75% ferrosilicon	365
Secondary aluminium in ingots	56.5
Burnt lime	200
Fluorspar	25
Graphitised electrodes	7.7
Coke gas, m ³	700
Electric power, kWh	800

Chapter 8

PRODUCTION OF FERROVANADIUM

Use of Ferrovanadium

Vanadium is one of the elements which possess excellent deoxidising properties. It is also used in the production of special steels. Because of its ability to increase the strength and improve the plastic properties of steel, as well as to raise the latter's resistance to attrition and impacts, vanadium is widely used in the production of various structural, tool and spring steels.

As a rule, vanadium is introduced in combination with chrome, nickel, molybdenum, tungsten and other elements.

Compounds of vanadium are widely used in various fields of national economy: chemical industry, agriculture, medicine, textile industry, lacquers and paints, etc. The bulk of vanadium output is consumed by ferrous metallurgy in the form of ferrovanadium, the composition of which, as specified by ГОСТ 4760-49, is given in Table 61.

Table 6

Chemical Composition of Ferrovanadium (ГОСТ 4760-49)

Grade	Composition, %						
	V minimum	C	Si	P	S	Al	As
B1	35	0.75	2.00	0.10	0.10	1.0	0.05
B2	35	0.75	3.00	0.20	0.10	1.5	0.05
B3	35	1.00	3.50	0.25	0.15	2.0	0.05

Grades B1 and B2 ferrovanadium are used for alloying the most crucial grades of steels; B3 ferrovanadium, grade with a higher phosphorus content, is used for alloying pig iron and low-quality steels.

Physicochemical Properties of Vanadium

Vanadium (V) is a chemical element of the fifth group of the Mendeleev Periodic Table of Elements. Vanadium was discovered in Mexico in 1801 but until 1830 was mistaken for chrome. Metallic

vanadium was obtained in 1869 by the action of hydrogen on vanadium chloride with simultaneous heating.

Its content in the earth's crust amounts to 0.015%. The assay of vanadium in the earth's solid shell is very low.

Vanadium is a brittle, very hard metal, light-silvery in colour, with the following physicochemical properties:

Atomic weight	50.95
Specific gravity	6.11
Melting point, °C	1900
Boiling point, °C	3000
Valence	2 to 5

Vanadium is stable in dry air at room temperature and oxidises rapidly at high temperatures.

It is 100% soluble with iron in all proportions in liquid as well as in solid states. An alloy with 31% V has the minimum melting point of 1440°C.

With carbon vanadium forms a number of carbides. The system has not been studied sufficiently. Vanadium carbides are very stable compounds and their melting points range from 2400 to 2800°C, depending on the composition.

With silicon vanadium forms silicides VSi_2 , V_3Si and, presumably, V_2Si .

With sulphur vanadium forms sulphides V_2S_5 , V_2S_3 and VS . With oxygen it forms a number of oxides: basic oxides VO , V_2O_3 , amphoteric oxide VO_2 and vanadium pentoxide V_2O_5 , an amphoteric oxide with the predominance of acid properties.

The formation of vanadium oxides from elements is accompanied by an evolution of heat. The stability of vanadium oxides increases with the decrease in the oxygen content per unit of vanadium.

Vanadium pentoxide VO is a greyish powder with specific gravity of 5.5 and a melting point of 1970°C.

Vanadium trioxide V_2O_3 is brilliant black in colour, its specific gravity is 4.84 and its melting point 1967°C. When heated V_2O_3 readily oxidises in the air to VO_2 .

Vanadium dioxide VO_2 is bluish-black in colour, its specific gravity is 4.3 and its melting point 1545°C.

Pentoxide V_2O_5 is a powder, orange-yellow or brick-red in colour, with specific gravity of 3.32, melting point of 675°C and boiling point of 2245°C. Vanadium pentoxide serves as a raw material for the production of vanadium and ferrovandium and is also widely used as a catalyst in the manufacture of sulphuric acid and other chemical products.

Although the vanadium content in the earth's crust is high, vanadium is considered to be a rare element as it is very much dispersed in nature.

It is a component of iron titan-magnetite ores. The approximate composition of these ores is given in Table 62.

Table 62

Composition of Vanadium-Bearing Iron Ores

Deposit	Chemical composition, %					
	Fe ₂ O ₃	FeO	V ₂ O ₅	TiO ₂	SiO ₂	Cr ₂ O ₃
Kusinskoye . .	34-38	28-31	0.63-0.68	13-14	4-10	0.5-0.7
Pervouralsk . .	32-36	21-23	0.5-0.60	4	12-16	0.1-0.2
Kerch	Fe 38	—	V 0.07- 0.08	—	14	—

Deposit	Chemical composition, %					
	Al ₂ O ₃	MnO	CaO	MgO	P	S
Kusinskoye . .	4-8	0.2-0.3	2-3	4-5	0.01-0.02	0.1
Pervouralsk . .	10-12	0.2-0.3	5-7	6.5-7.5	0.01-0.02	0.04-0.08
Kerch	8	5	—	0.4	1.0	0.2

Technology of Ferrovanadium Manufacture

The vanadium content in ores is low and for this reason the manufacture of vanadium alloys is a combination of a number of metallurgical and chemical processes.

Vanadium-bearing ore is used for smelting pig iron with 0.45-0.54% V. The next stage is the transfer of vanadium from pig iron to the slag in an open-hearth furnace or, as a rule, by processing pig iron in a converter. Approximately 90% V contained in the pig iron is thus reverted to the slag. Vanadium in the slag is in the form of FeO·V₂O₅.

Evaluated in V₂O₅, the summary content of V₂O₃ and V₂O₅ averages 12-14% in converter slags. In some instances it is worthwhile to recover vanadium directly by chemically processing the ores, without smelting vanadium-bearing pig iron.

To recover vanadium from the slag it is necessary to turn it into compounds soluble in water or in diluted solutions of acids. To this

end the converter slag is crushed and milled in ball mills; the metallic particles are then eliminated by magnetic separation. The transfer of vanadium to soluble compounds is effected by the oxidation roasting of the slag at 770-820°C with alkaline additions. Sodium chloride and silvinit (NaCl+KCl) are used as alkaline additions, the amount averaging 10-12% of the weight of the slag. The mixture is roasted in rotating kilns.

As a result of roasting the bulk of V_2O_3 oxidises to V_2O_5 and combines with alkaline additions, giving compounds soluble in water—sodium vanadates of the $Na_2O \cdot V_2O_5$ type. Roasted slag is leached in special installations and insoluble particles are filtrated.

The filtrate, containing sodium and potassium vanadates, is acidified with sulphuric acid and heated. This results in the precipitation of technical vanadium pentoxide which contains 80-90% V_2O_5 . The precipitate is filtrated, dried and melted in dry-hearth furnaces at 700-750°C. Liquid vanadium pentoxide is poured onto a massive iron disc where it solidifies in the form of thin platelets.

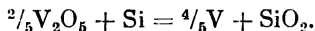
The recovery of vanadium from the slag averages 80-85%. Melted technical vanadium pentoxide has approximately the following composition: 85% V_2O_5 , 2% SiO_2 , 6% Fe_2O_3 , 1% Cr_2O_3 , 1% Al_2O_3 , 1.5% MnO , 1% CaO , 0.7% MgO , 0.07% P; the rest is $Na_2O + K_2O$.

Smelting of Ferrovanadium from Vanadium Pentoxide

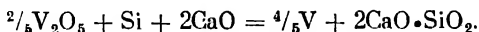
Ferrovanadium may be obtained by the reduction of vanadium oxides by carbon, silicon and aluminium.

The manufacture of ferrovanadium by the reduction of its oxides by carbon has not found wide application because vanadium is a carbide-forming element and the resulting alloy, containing 4-6% C, cannot be used for smelting low-carbon steels, for which ferrovanadium is chiefly used. Consequently, the bulk of ferrovanadium is produced by the reduction of vanadium pentoxide by silicon and aluminium.

The reduction of vanadium pentoxide by silicon proceeds as follows:



The lower oxides V_2O_3 and VO formed in the course of the above-mentioned process attack silicon-giving vanadium silicates, which hamper the reduction process. To prevent that, lime is introduced into the mixture; this bonds silicon into stable compounds and hinders the formation of vanadium silicates. In the presence of calcium oxide the reduction reaction takes the following form:



The reduction of vanadium pentoxide by silicon requires high temperatures and the silicothermic process of ferrovanadium manufacture is, therefore, effected in closed electric furnaces of the steel-smelting type. Roof, hearth and walls of the furnace are lined with magnesite brick. The heats are run with the secondary voltage at 210-220 V and power input at the transformer is 1,500 kVA. The furnace is equipped with graphitised electrodes, this reducing the transfer of carbon to the alloy.

The mixture materials for the production of ferrovanadium by the silicothermic method are melted vanadium pentoxide, 75% ferrosilicon, soft iron cuttings and lime. Smelting is done in two stages: reduction and refining.

In the first period the estimated amount of iron cuttings is charged into a well-heated bath, power is applied and, as the iron melts down, a mixture of vanadium pentoxide with lime and ferrosilicon is added, the amount of ferrosilicon being in excess of the amount required. As soon as the mixture has melted down, the heat is killed to let the processes come to completion. Following the delay, the slag is drained. The silicon content in the alloy at the end of the reduction period reaches 15%, while the slag usually averages 50-55% CaO, 5-10% MgO, 28-30% SiO₂, 0.5% V.

After the slagging-off, the alloy is refined from silicon; this requires the charging of vanadium pentoxide and lime. The vanadium pentoxide contained in the slag attacks the alloy silicon, and as a result of that vanadium is reverted to the slag, while the silicon content in the alloy decreases.

At the end of the refining period the alloy is sampled; the sample fracture permits to ascertain the silicon content in the alloy.

The final slag contains 40-45% CaO, 20-25% SiO₂, 10-15% MgO, 6-12% V₂O₅. This slag is returned to the furnace in the reduction period of the next heat.

A negligible amount of aluminium is added in the reduction period to promote full reduction of the vanadium contained in the slag.

The approximate consumption of raw materials per ton of 40% ferrovanadium in an electric furnace is as follows, kg:

Melted vanadium pentoxide	800
75% ferrosilicon	400
Secondary aluminium	90
Lime	1,300
Iron cuttings	370
Graphitised electrodes	30
Electric power, kWh	1,500
Recovery, %	98.9%

The over-all recovery of vanadium from ore to ferrovanadium averages approximately 54%.

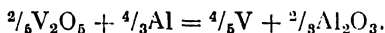
When organising production control emphasis should be laid on the leaching and precipitation of vanadium pentoxide from its solution.

The vanadium content in the slags should be carefully controlled when smelting ferrovanadium in electric furnaces. Intermediate slagging-off is permitted provided the vanadium pentoxide content in the slag is below 0.5%.

Ferrovanadium is teemed into vertical moulds and, when it has been cooled, dressed and packed in cases or metallic drums. Steps should be taken to cut short alloy dressing losses. Siftings obtained in dressing and packing of ferrovanadium (slagged alloy lumps and metallic particles) are resmelted.

Melters should concentrate on eliminating vanadium losses in the production of ferrovanadium, as a low vanadium assay in the initial ores and its complicated winning methods are the cause of its high cost.

In the aluminothermic process the reduction of vanadium pentoxide is described by the reaction:



The reduction of vanadium oxides by aluminium is sufficiently complete.

When reducing pure V_2O_5 by aluminium the evolution of heat amounts to 1,038 cal per kg of initial materials. The composition of the mixture is as follows, kg:

Melted pentoxide	100
Aluminium	46.1
Iron cuttings	71.2

The fusing crucibles used for the smelting process are lined with magnesite brick or manufactured entirely from metal. The crucible is mounted on a carriage. The smelting method requires the use of bottom priming with a uniform charging rate equal to the melting rate. Excessive charging may cause a violent flow of the process, accompanied by eruptions. Slow charging is the cause of the cold run of the furnace, and that results in lower vanadium recovery to the alloy. The recovery of vanadium in this process comes to 92-95%.

Reduction of vanadium pentoxide is sometimes effected in closed electric furnaces to bolster vanadium recovery; after the consumption of the initial process power is applied and the remaining

vanadium pentoxide is exposed to reduction by aluminium and carbon for 1.5-2 hours, after which the alloy and slag are tapped.

The aluminothermic method is capable of producing commercial vanadium (up to 97%) by the reduction of pure vanadium pentoxide by aluminium with the addition of calcium oxide or fluor-spar as a flux. Pure vanadium pentoxide is, in turn, produced by the chemical processing of melted vanadium pentoxide.

Pure vanadium is produced by the vacuum reduction of vanadium oxides by calcium or carbon. The metal produced by this method contains 99.3-99.5% V and is remarkable for high plasticity.

Chapter 9

FERRONIBIUM

Use and Composition of Ferroniobium; Raw Materials

The addition of niobium to stainless and heat-resisting steels enhances their plasticity and corrosion resistance. The introduction of niobium in structural steels improves their welding properties, increases the strength and plasticity of steel and prevents the corrosion of the welding seams. Niobium is also introduced into steels with special physical properties.

The composition of ferroniobium used for alloying steel is given in Table 63.

Table 63

Composition of Ferroniobium (as Specified by MILITARY 2735-51)

Grade	Chemical composition, %						
	Nb-Ta minimum	impurities, maximum					
		C	Si	Al	P	S	Ti
1161	50	0.12	10.0	7	0.27	0.03	7
1161	50	0.20	11.5	7	0.32	0.05	7

Niobium is a comparatively rare element, its content in the earth's crust averages 0.001%. As a rule, it is accompanied by tantalum and, consequently, one always has to deal with niobium-tantalum ores.

The most common minerals are the columbite (columbium is another name for niobium) $(\text{Fe, Mn})\text{O}(\text{Nb, Ta})_2\text{O}_5$ —a complex columbite, and tantalate of iron and manganese—and pyrochlore $(\text{Na, Ca})_2 \cdot (\text{Nb, Ta, Ti})_2 (\text{O, F})_7$.

Niobium is also contained in the mineral loparite which averages 10% Nb_2O_5 .

The niobium concentrate, a mixture of minerals resulting from ore beneficiation, is used as raw material for the production of fer-

roniobium. The main niobium-bearing mineral in the concentrate mentioned is the pyrochlore. According to technical specifications, the niobium concentrate assay should meet the requirements listed in Table 64. In addition to the admixtures mentioned in the table, the niobium concentrate contains about 12% ZrO_2 .

Table 64

Composition of Niobium Concentrate
(as Specified by MINT 4423-54)

Grade	Chemical composition, %, absolutely dry matter						Moisture, % not more than
	(Nb+Ta) ₂ O ₅ minimum	impurities, %, to (Nb+Ta) ₂ O ₅ , %, ratio, maximum					
		P	SiO ₂		C	TiO ₂	
KII	37.0	0.003	0.32	0.005	0.005	0.35	1.0

Technical niobium pentoxide, obtained by chemical processing of niobium concentrate, is used along with niobium concentrate for smelting ferroniobium. The flow sheet of the process is shown in Fig. 86.

Physicochemical Principles of Ferroniobium Production Process

Elemental niobium is a plastic metal, grey in colour, with the following chief physicochemical properties:

Atomic weight	92.91
Specific gravity	8.57
Valence	2, 3, 4, 5
Melting point, °C	2415
Boiling point, °C	3300
Heat of fusion, cal	4,773
Specific heat capacity at 20°C, cal/°C	0.0645

With iron, niobium gives a compound Fe_3Nb_2 which may dissolve a great amount of iron but not niobium. The melting point of commercial alloys, with 50-60% Nb, is equal approximately to 1600°C.

With carbon, niobium forms carbides of varying composition from NbC to Nb $_4$ C. The melting point of NbC is 3500-3800°C. Hardness of niobium carbides is 9-10 Mohs which determines its use in the cemented carbide tool alloys.

With oxygen, niobium forms three oxides: niobium pentoxide Nb $_2$ O $_5$, niobium dioxide NbO $_2$ and niobium protoxide NbO. Nio-

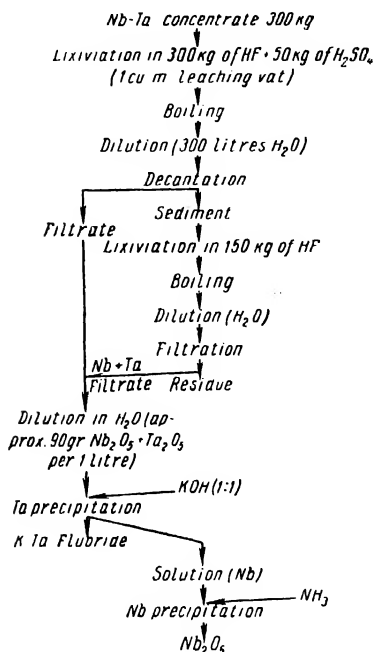


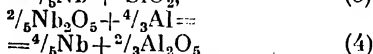
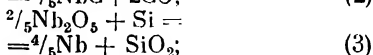
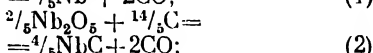
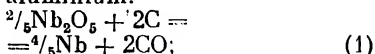
Fig. 86. Flowsheet of niobium pentoxide production

niobium pentoxide Nb_2O_5 is an acid oxide and a fine crystalline powder, white in colour, with specific gravity of 4.55 and a melting point of 1460°C .

Niobium dioxide NbO_2 is a powder, blue-black in colour; it is unstable and decomposes to Nb_2O_5 when heated. The most stable oxygen compound of niobium is the monoxide NbO . It is a black powder with specific gravity of 6.27.

The character of niobium oxides varies from acid to basic with a decrease in the amount of oxygen linked to niobium.

Niobium pentoxide, as pointed out by Prof. V. P. Yelyutin, may be reduced by carbon, silicon and aluminium:

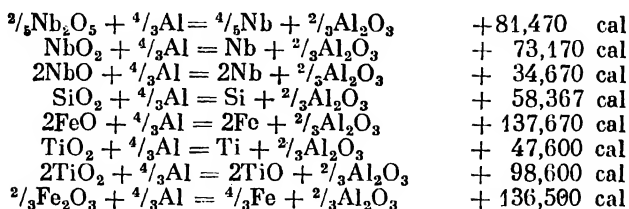


Calculations show that these reactions are quite feasible and that the theoretic temperatures for the beginning of the reactions (1) and (2) are 1100°C and 890°C respectively. Ferroniobium, however, is not manufactured with the use of carbonaceous reducing agents, for it is used for alloying steel with a low-carbon content and its saturation with carbon in the reduction reactions (1) and (2) is unavoidable.

When niobium pentoxide is reduced by silicon, incomplete reduction reactions occur along with its reduction to metal (with niobium reduced from pentoxide to lower oxides). The reduction of lower oxides by silicon is associated with considerable difficulties and leads to the formation of silicon-bearing alloys and high niobium slag losses.

The highest niobium recovery is attained with the use of aluminium as a reducing agent. Therefore, the aluminothermic method is employed commercially for the production of ferroniobium.

The reactions of ferroniobium smelting by the aluminothermic method are as follows:



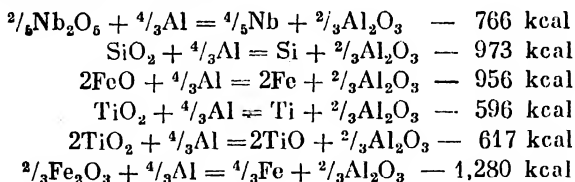
The remaining oxides are reduced at the same time.

The following recovery coefficients (%) from mixture to the alloy may be assumed in charge calculations on the basis of operating practice:

Niobium	98 (3% in the form of prills)
Silicon	80
Titanium	45
Iron	99
Zirconium	7

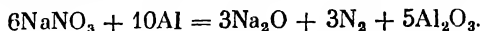
In ferroniobium smelting the minimum specific heat capacity of the process is equal to 720-740 kcal per kg of mixture.

As a result of reduction the following amount of heat is generated per kg of oxide reduced:



Consequently, the specific heat of the process equals approximately 400 kcal per kg of mixture.

An additional amount of heat is obtained as the result of oxidation of a supplementary amount of aluminium by saltpetre:



Oxidation of aluminium by one kg of saltpetre generates 3,132 kcal of heat.

When calculating the charge the aluminium content in the alloy is assumed to be equal to 5.6%, while that of niobium—62%; the

necessary additional amount of iron is introduced in the form of iron ore.

Table 65

Composition of Charge Materials for Ferroniobium Smelting

Material	Chemical composition, %													other impurities
	(Nb+Ta) ₂ O ₅	SiO ₂	ZrO ₂	TiO ₂	FeO	C	S	P	Al	Fe	Si	Fe ₂ O ₃	NaNO ₂	
Niobium concentrate	40.0	12.0	12.0	8.0	1.5	0.12	0.13	0.10	—	—	—	—	—	—
Aluminium powder	—	—	—	—	—	—	—	—	97.0	0.8	0.2	—	—	2.0
Iron ore	—	1.5	—	—	—	—	0.04	0.03	0.5	0.3	0.2	97.5	—	—
Saltpetre	—	0.3	—	—	—	—	—	—	—	—	—	—	98.0	1.7

When calculating the charge per 100 kg of concentrate and proceeding from the assay of materials given in Table 65, the following calculated composition of the charge is obtained, kg:

Niobium concentrate	100	Iron ore	8.2
Aluminium powder	43.8	Sodium saltpetre	24.8

A charge table drawn up on the basis of calculations and smelting practice (Table 66) is usually used for calculating the composition of the charges.

In producing ferroniobium from siftings obtained by beneficiation of rich slag from ferroniobium smelting, Table 67 is used for working out the charge.

Table 66

Practical Table for Working Out the Charge for Smelting Ferroniobium from Pyrochlore Concentrates

(Nb+Ta) ₂ O ₅ content in the concentrates, %	Amount of mixture materials, kg			
	concentrate	aluminium powder	saltpetre	iron ore
50-48	100	46-45	24-23	10-9
48-46	100	45-44.5	24-23.5	10-9
46-43	100	45-44	25-24	9.5-9
43-40	100	44.5-44	25.5-25	9-8.5
40-37	100	44-43	26-25.5	8.5-8

Table 67

**Table for Working Out the Charge for Smelting
Ferroniobium from Sifting**

Assay of the metallic fraction in the siftings, %	Amount of mixture materials, kg		
	siftings	aluminium powder	saltpetre
10-15	500	215-205	300
15-25	500	180-170	250
25	500	145-135	200

When commercial niobium pentoxide is used, the charge is composed as follows, kg:

Commercial niobium pentoxide	100
Aluminium powder	48.5
Iron ore	18-20
Sodium saltpetre	11-13

Smelting Technology

The concentrate is thoroughly dried and milled into particles less than 2 mm in size. The aluminium powder is prepared from virgin aluminium; its sizing should be as follows:

Size	< 0.1 mm	0.1-1.0 mm	1.0-3.0 mm
Content, %	< 10	> 80	< 10

Hematite ore containing from 67 to 69% of iron is dried and sieved through a 2-mm sieve.

Saltpetre should be white in colour and contain approximately 98% NaNO_3 . Before being used it is dried and sieved through a 2-mm sieve.

The charge is carefully weighed and mixed in a drum mixer for approximately 30 minutes.

Ferroniobium is smelted in sectional cast-iron shafts with 50 to 60 mm side walls. The shaft is installed on a carriage and the seams are luted up with refractory clay. The bottom is then tamped with a layer of approximately 150 mm of magnesite powder, upon which a course of magnesite brick-on-the-flat is laid.

The inside of the shaft is lined with magnesite brick and the seams are filled with magnesite powder to prevent carbon pick-up from the cast-iron shaft.

The smelting of ferroniobium is practised with bottom priming. The heat lasts approximately one minute if the charge weighs 1,500 kg.

The heat is kept in the shaft for not less than three hours, after which the shaft is dismantled while the alloy ingot is kept in the car for another three hours. When it has been cooled, the ingot is dressed: slag is conveyed to a special slag dump (its mixing with the slag from other processes is to be avoided), while the alloy is crushed into lumps of up to 10 kg in weight, cleaned and packed in metallic drums.

The chief efficiency factor in ferroniobium smelting is the recovery of niobium, for its cost comes to approximately 92% of that of alloy manufacture.

At present, recovery of niobium fluctuates around 90% with raw materials consumption per basic ton of alloy (50% Nb) being as follows, kg:

Niobium concentrate	2,500
Primary aluminium	930
Iron ore	170
Sodium saltpetre	750

Chapter 10

ZIRCONIUM FERROSILICON AND FERROBORON

Zirconium Ferrosilicon

Zirconium is used in metallurgy because it is an excellent steel deoxidiser. Moreover, zirconium paralyses the noxious effect on steel by nitrogen and sulphur by bonding them into stable compounds.

Zirconium is added to steel in the form of zirconium ferrosilicon which, according to technical specifications, should not contain less than 40% of zirconium and have the $\frac{\text{Si}}{\text{Zr}}$ ratio of not over 0.55 and the $\frac{\text{Al}}{\text{Zr}}$ ratio equal to 0.20.

It is quite a widely occurring element, its content in the earth's crust being equal to 0.02%; zirconium ore reserves in the U.S.S.R. fully ensure industrial needs in zirconium and its alloys.

Zirconium possesses the following chief physicochemical properties:

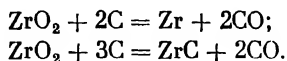
Atomic weight	91.2
Specific gravity	6.5
Valence	2.4
Melting point, °C	2000

With iron, zirconium gives a stable compound Fe_2Zr ; with carbon, zirconium gives a stable carbide (with composition varying from ZrC to Zr_3C), which is unavoidably present when zirconium is reduced by carbon. Zirconium gives a number of silicides with silicon: Zr_4Si , Zr_2Si , Zr_3Si_2 , Zr_4Si_3 , Zr_6Si_5 , ZrSi and ZrSi_2 . Zirconium forms three oxides with oxygen: ZrO_2 , Zr_2O_3 and ZrO . Zirconium oxide ZrO_2 is the best known; it is an amphoteric oxide, being a powder white in colour, with a melting point of 2800°C.

Zirconium occurs in nature in the form of various minerals, among which commercially important are zircon, baddeleyite and eudialyte. Most wide-spread is the mineral described by the formula $\text{ZrO}_2 \cdot \text{SiO}_2$, named zircon.

A concentrate with the following composition: 58-66% ZrO_2 , 31-38% SiO_2 , 1-4% Fe_2O_3 and up to 5% TiO_2 , may be obtained as a result of beneficiation.

When reducing zirconium dioxide by carbon two parallel reactions occur

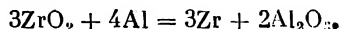


The temperature at the beginning of the above reactions is respectively 2170°C and 1500°C, which testifies to the predominant flow of the second reaction; thus in reducing zirconium oxides by carbon it is impossible to obtain a low-carbon, low-silicon alloy.

The presence of silicon hampers the formation of zirconium carbides and zirconium ferrosilicon is, therefore, smelted in order to produce an alloy low in carbon.

Low-percentage zirconium ferrosilicon (up to 30% Zr) may be smelted by a continuous process in an electric furnace with carbon-reducing agents. In this instance, the charge is made up of zirconium concentrate, quartz and charcoal.

Silicon may be introduced into the alloy in the form of 90% ferrosilicon; when this is done quartz is not added to the charge, while the amount of carbon is calculated for a complete reduction of zirconium oxides and iron. Lime is used as flux. This method may give an alloy with 40-45% Zr, 40-45% Si and 0.2% C. The recovery of zirconium then amounts to 50-60%, electric power consumption is 15,000 kWh per ton of alloy with a mean zirconium content of 45%. Aluminium may be used as a reducer; in this case the reduction reaction is as follows:



Ferroboron

Addition of boron to steel has become common practice. This enhances the mechanical properties and hardenability of steel. Boron is added to steel in negligible amounts (0.0025-0.0030%) in the form of ferroboron and ferroboral. Moreover, boron is employed in metallurgy as a strong deoxidiser.

According to existing technical specifications Soviet industry produces ferroboron of the following composition (grade B1): $\geq 5.0\%$ B, $\leq 3.0\%$ Si, $\leq 5.0\%$ Al, $\leq 0.25\%$ C.

The composition of ferroboral is given in Table 68.

Boron occurs widely in nature, its content in the earth's crust being 0.01%. The most common minerals of boron are boracite $\text{Mg}_2\text{Cl}_2\text{B}_{10}\text{O}_{30}$ and ascharite MgHBO_3 . Reserves of borates available in the Soviet Union meet fully the requirements of the national economy.

Table 68

Ferroboral

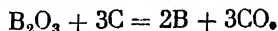
Grade	Chemical composition, %		
	B	Si:B	Al:B
BA1	4.0	1.0	1.3
BA2	3.6	1.3	1.5

Boron is characterised by the following physicochemical properties:

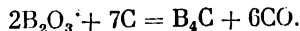
Atomic weight	10.82
Specific gravity	2.3
Melting point, °C	2300
Valence	3 and 5

With iron boron gives iron borides Fe_3B and FeB . With carbon, boron forms boron carbide B_4C with hardness almost equal to that of a diamond and with a smelting point of 2400-2450°C. With aluminium, boron forms aluminium borides the composition of which is AlB_2 and AlB_{12} . With oxygen, boron gives a number of oxides: B_2O_3 , B_4O_5 , B_2O_2 , and B_4O_3 .

Ferroboron may be produced by the reduction of boron oxides by carbon in the presence of iron in an electric furnace. The reduction of boron oxides by carbon may be represented by the reaction

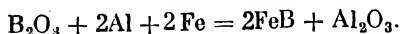


The theoretical temperature for the initiation of this reaction is 1836°C, but it is more probable that the reduction of boron stops at boron carbide



As boron carbide is a very stable compound, alloys produced by the above method unavoidably carry some carbon.

The reduction of boron by aluminium is facilitated by the presence of iron because it helps form iron boride FeB which permits obtaining an alloy low in aluminium. The reaction of boron reduction is



The aluminothermic process is at present the main method of ferroboron production. The mixture is approximately of the following composition, kg:

Borate ore	2,016
Iron ore (hematite)	1,350
Aluminium (powder)	710

Boron recovery to the alloy averages approximately 50%.

Rich ferroboration with 13-14% B and 5-6% Al (boron recovery being 58-65%) may be produced with potassium chlorate as a thermit addition.

It is worthwhile to smelt ferroboration in an arc furnace with aluminium acting only as a reducer (no aluminium is provided for the reduction of thermit additions) as long as the lack of heat is compensated from electric power sources. Ferroboration with 12-13% B, 3-4.5% Al and 3-4% Si and with boron recovery of up to 50%, has been produced in trial heats. Saving of aluminium, as compared to the shaft-smelting method, amounts to 36-40%.

Part III

PRODUCTION METHODS

Chapter 1

COMPOSITION AND LAYOUT OF SMELTERS

The composition and type of smelter shops, organisation of raw-material supplies and the works layout are determined by a plant's productivity and the planned grades of alloy output. For example, if a plant produces only silicon or chrome alloys, one common stock yard will suffice even if there are several smelter shops. On the other hand, when smelting several different alloys, even in one smelting shop, two separate stock yards should be organised to ensure the purity of raw materials.

The plant should include a complex of auxiliary shops: mechanical, electrical, transport, laboratory, etc., which are necessary for the continuous operation of smelter shops, spare parts supplies, repairs and delivery of materials. It should be pointed out that deliveries are an integral part of the technological processes and their uninterrupted operation is imperative for the normal flow of the production process. The plant should be provided with necessary and appropriately equipped warehouses and sites to store minimum reserves of raw and auxiliary materials (refractories, sand, clay), in the absence of which normal plant operations are impossible.

Workshops for the manufacture of electrode shells and tares should be provided for in the plant layout.

Lime-burning shop is to be included if the plant produces alloys with lime as a flux. Power supply of the works is ensured either directly by an electric power station or by substations and electric systems.

The cooling systems of furnaces and granulation installations use water. Water supply should be faultless as an interruption may cause serious failures.

Part of the water is usually circulated (to economise on fresh water supply) through sprinkling basins or cooling towers, and then returned to the furnaces.

Water is passed through settlers and chemically purified to prevent the water-cooling systems from scaling. These operations are performed by the water-supply workshop.

The arrangement of separate shops depends on local conditions. It should ensure the shortest possible routing and conform with sanitary and fire-hazard regulations.

Special consideration should be given to the rational laying of railway tracks, which should have the minimum of curves, crossings and inclines. All crossings should be provided with flooring and equipped with lifting gates where traffic is heaviest.

The work site should be provided with a drainage system, covered with asphalt or paved, lighted and planted with trees or shrubs.

Composition and Layout of a Ferroalloy Smelting Shop

The chief factors to be thoroughly weighed prior to the choice of the type of ferroalloy shop and its layout are:

- 1) planned output;
- 2) character and amounts of materials to be stored;
- 3) pre-smelting preparation of raw materials;
- 4) handling of prepared charge materials, their storing and proportioning;
- 5) charging methods;
- 6) teeming methods;
- 7) handling of ferroalloys and slags, dressing and storing of ferroalloys;
- 8) appropriate working conditions.

In accordance with the above a ferroalloy shop should have the following main sections:

- 1) stock yard;
- 2) materials preparation section (it may be part of the stock yard);
- 3) depot for prepared mixture and its proportioning (usually situated in the smelter building, though it may be part of the stock yard);
- 4) smelter shop;
- 5) slag yard;
- 6) alloy dressing and storing floor.

Stock yard. When selecting the site for the stock yard one must aim at minimising the route of the materials from the stock yard to the furnaces; besides, one should provide for the possibility of future expansion not only of the smelter shops, but the stock yard. This problem may be solved in a number of ways. In one instance,

the smelting shop and the stock yard are situated parallel to each other and the materials are conveyed through a perpendicular inclined gallery. This is convenient when supplying a number of smelter shops from one stock yard.

In another instance, the materials depot, mixture preparation section and smelter shop are arranged in a row so that all materials, from raw stock to finished ferroalloys, move along a line parallel to the axis of the building. Such a layout considerably increases the route of the materials inside the shop.

In shops smelting ferrotungsten, ferromolybdenum, etc., where the volume of production and, consequently, the need in raw materials is relatively low, the stock yard is arranged as an aisle of the smelter shop, although this impairs the ventilation of the smelter shop.

A stock yard should be sheltered. In rigorous climatic conditions, it should have heated floor space for mixture preparation.

Either of the following two types of stock yards is used in ferroalloy works: the bin type, with underground bins, in which railway cars come to the bins and the material is discharged below onto a belt conveyor through special bin gates; the bedding type, with beds situated below the floor level (railway tracks are situated on the floor level), where further material conveyance is ensured by overhead bucket cranes.

The first type of storage is somewhat more convenient to operate, though considerably more expensive to build.

The volume of bins for storing raw materials is determined by shop productivity, the distance of the plant from the sources of supplies and periodicity of their delivery.

The bins should be so situated as to preclude the possibility of mixing materials in the course of unloading and conveying operations. Usually, the stock yard is a two-aisle structure.

The construction and equipment of the mixture preparation section are determined by the diversification of the alloys smelted and by the shop's output volume. For example, the burden preparation section in a ferrosilicon shop has two routes: for quartz and for coke.

The former is equipped with washer drum, crushers (preferably of cone type) and screens.

The latter includes roll crushers, vibrating screens and a drying kiln for drying fine coke to a predetermined moisture content.

A special crusher should be provided for handling metal turnings.

All mixture preparation operations should be mechanised and, wherever possible, automated.

Raw-material conveyance to the smelting shop may be solved by belt conveyors, elevators, telephers or skip hoists, depending on the character of the materials handled. Generally used are belt

conveyors, installed in a housed gallery (two conveyors should be made available for continuous operations).

A ladle hoist may be used successfully, this being a simple and reliable mechanism.

Smelter shop. A smelter shop consists of transformer, furnace and teeming aisles (Fig. 87).

The transformer aisle of a smelter shop is a 6-m wide four-storey structure. The furnace transformers, oil-coolers and oil breakers are situated on the ground floor at the +1.0-m reference mark; control panels, rest rooms, etc., are installed on the first floor at the +5.0-m level; the second floor, between the +13.0- and +20.0-m marks, is occupied by mixture bins (proportioning bins); the horizontal conveyors which distribute mixture materials to the bins (the materials from the stock yard are conveyed by an inclined conveyor) are on the third floor.

Raw materials are dumped into metallic or concrete bins from horizontal conveyors by trippers. The proportioning bins should hold at least 24 hours' supply; this helps to organise one-shift operation of the stock yard. The bins are either metallic or concrete. Their gates are equipped with vibrating feeders powered by electro-mechanical drives.

The mixture is conveyed from the proportioning bins to the furnace hoppers by means of a proportioning monorail grab bucket crane. The latter is equipped with weighing scales and provided with push-button control and a device which opens the bucket over the required furnace hopper.

The installation of the transformer on the working-floor level, as well as its disposition to the furnace aisle, allows to shorten the low-voltage circuit and cut down electrical losses.

When the transformer is installed on the furnace floor the transformer aisle is unnecessary, for the furnace can be charged with the help of an overhead electric crane equipped with bottom-drop buckets, which are conveyed with charge portions from under the proportioning bins. The proportioning bins are located in the stock yard which, in this case, is usually contiguous with the smelter shop. It is recommended to convey mixture batches with the aid of skip hoists onto a monorail crane which transports them directly to furnace hoppers.

The furnace aisle consists of several storeys, which house furnaces, working floors and platforms for furnace servicing, electrode jointing, mixture proportioning and electrode hoisting winches.

The furnace foundations are at shop-floor level. The working floor intended for furnace-top operation is situated at the +4.5-5.0-m level; furnace hoppers, working decks for anode paste discharge and electrode shell joint-making, as well as the working platforms

for ventilation fans and, sometimes, electrode positioning winches, are arranged on the second floor, at approximately the +13.0-15.0-m level. In some cases the winches are installed on a special platform at +20.0-m level.

Sometimes overhead cranes, used for electrode jointing, reclamping and furnace charging, are made available over the electrode platforms.

To ensure unhampered operation of furnace mechanisms and provide normal working conditions the width of the working platforms should not be less than 18 m for larger furnaces and 15 m for smaller ones.

Working platforms are provided with balconies projecting over the teeming aisle; they serve as receiving platforms for reverts (for resmelting) and the various materials and assemblies required for furnace repairs are conveyed by overhead cranes.

The interval between the furnace axes should be minimum 18 m for smaller furnaces and 30 m for larger ones; the latter interval may be reduced to 24 m when closed-top furnaces are used.

An electric hoist is provided to convey anode paste and electrode shells to the electrode platform. Inside the shop anode paste is transported in power trucks or carriages. At present manual labour has been eliminated since the paste is transported by electric cranes in bottom-drop buckets.

Working platforms must be manufactured of non-conducting materials to provide for safer working conditions. Forced ventilation outlets should be available at each working point.

The smelting sections of heat-treating shops are equipped with smelting chambers or exhaust hoods installed over the melting shafts. The working premises should be provided with a ventilation system ensuring the complete evacuation of gases and dust generated in the smelting processes.

These sections are equipped with overhead electric cranes.

The teeming aisle occupies a considerable portion of shop-floor space, and is approximately 20 m wide and about 20 m high.

The aisle is equipped with overhead electric cranes of 10 to 20 ton lifting capacity, depending on the alloys smelted and the volume of furnace output. Railway tracks are laid through (or terminate at) the smelter aisle.

Teeming equipment and granulation tanks are installed in the aisle, while tanks for water-cooling metal ingots are provided in the heat-treating shops. Ferrosilicon, ferrochrome and a number of other alloys are teemed into flat cast-iron or steel moulds. Vacuum chambers for the vacuum pouring of alloys are installed in the teeming aisle.

The installation of diverse pouring machines for teeming alloys enhances technology and improves operation results.

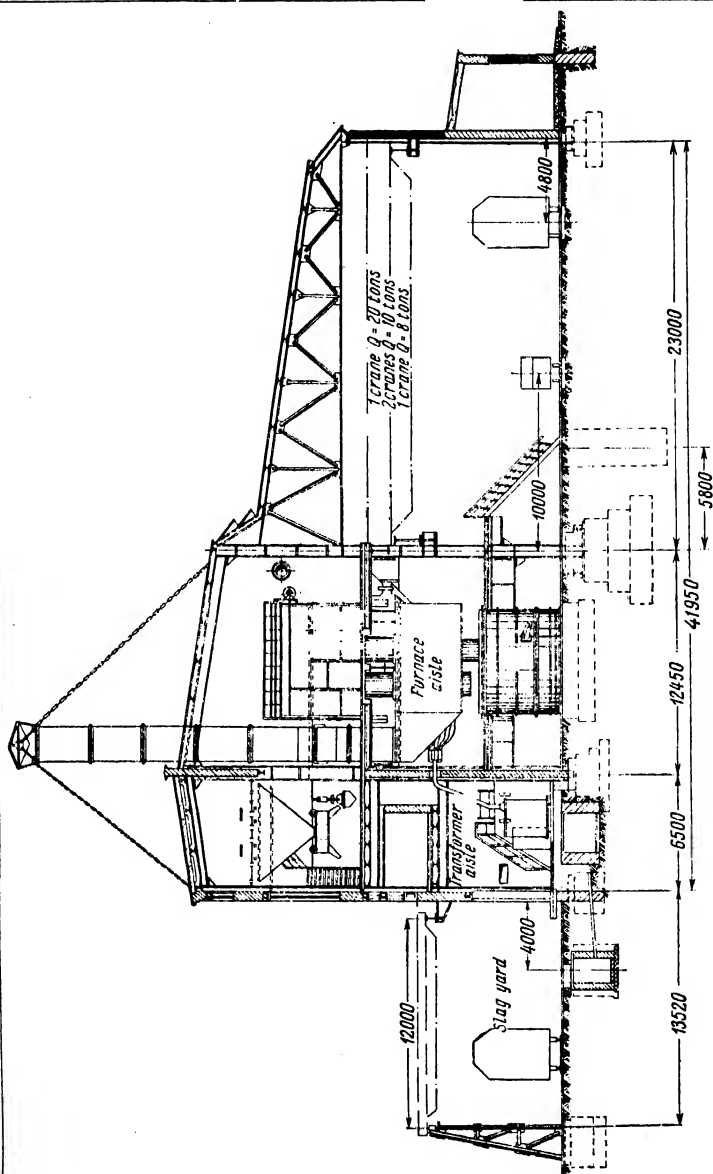


Fig. 87. Cross-section of a smelter shop

Space is allotted in the teeming aisle for storing current (one-day) supplies of refractories and for effecting repairs of pouring equipment.

A slag yard is usually attached to a smelter shop and is used for slag cooling and shipping. It is equipped with special devices for the recovery of metallic prills, scrap, etc. A simple example of such a device is a grate (with bins for dust) which serves to recover ladle crusts that form in the ladles in the process of teeming the refined ferrochrome. Ferrochrome slag separation branchworks are now being designed with a view to ensuring a more complete recovery of metal from the slag, as well as to make use of the latter as a building material. These branchworks must be situated near slag dumps.

The finished-products storage area should have sufficient floor space for ingot cooling (as it is used as a cooling area) and should be provided with devices for dressing and mechanised shipping of metal. Storehouses are usually equipped with primary and secondary crushers, pile drivers, cleaning drums and scales for weighing finished products.

Products are loaded into railway cars by electric self-propelled loaders.

The storehouse is usually a one-aisle structure (more often than not attached to the smelter building) and is equipped with electric overhead cranes.

Electric cars may be used as means of transportation between the various aisles of the shop, the finished-products storehouse and the slag yard. In some cases this may be effected by technological equipment: for example, metal is conveyed to the storage area by a pouring machine.

Smelter shops are provided with sanitary facilities and auxiliary premises such as recreation rooms, rest rooms, dining rooms, shower baths, installations for drying and dusting of working clothes, offices, etc.

It is recommended to situate auxiliary premises in a separate building, connected to the shop by a housed gallery; this ensures better aeration of the shop.

Chapter 2

LABOUR ORGANISATION AND WAGE SYSTEM

Labour and Production Organisation

A ferroalloy plant in the Soviet Union usually comprises a number of main and auxiliary shops, the so-called production links of the plant.

The main link of the enterprise is the smelter shop producing ferroalloys.

The shop superintendent is the head authority of the shop and is responsible for the over-all shop operation.

Relying in his work on the Party organisation, the shop public organisations and the permanent production conference¹, as well as making the most of the activity and initiative of the shop workers and submitting all important production problems for consideration by the entire shop, the superintendent should mobilise the efforts of the shop personnel for the fulfilment and overfulfilment of the quantitative and qualitative indices of the state plan.

The shop managerial staff also includes a deputy shop superintendent, maintenance assistant superintendent, rate-setting engineer, office manager and planning engineer.

Branchwork managers are in charge of different auxiliary sections (stock yard, cooling area, etc.).

The chief foreman controls production technology and equipment servicing and maintenance.

The chief foreman sees to it that his work team master and improve production technology, organises repair work and personally directs major technological operations (furnace heating after repair or shut-down, baking a new electrode, switching over a furnace from the smelting of one alloy to another, etc.).

The production of ferroalloys is a continuous round-the-clock process, effected on a shift basis.

A shift is headed by a shift foreman who directs and organises

¹ Production conferences in the U.S.S.R. are an important form of promoting the productive activity and creative initiative of the working masses. They were initiated in 1923 by Leningrad workers.

all the work during the shift; his orders are binding on the entire shift personnel.

Branchwork operations are in the direct charge of the foreman. He is endowed with the rights to ensure the fulfilment and overfulfilment of the state production plan. He is authorised by the shop superintendent to hire and discharge workers, set workers' rates in accordance with their qualifications, reward the foremost workers and punish those guilty of violating labour and technological discipline.

The instructions of the shop administration are transmitted to the workers only through the foreman, who is responsible for their implementation. The foreman has serious responsibilities. He must teach his men advanced methods of work, check on the observance of technological instructions and safety regulations, ensure high-quality production, inspect the equipment and its maintenance. The foreman is duty-bound to ensure one-hundred per cent fulfilment of the state plan.

There are two main forms of labour organisation—individual and team forms, depending on the character of the production processes and conditions of their realisation.

The individual form of labour organisation implies that the technological process is in the hands of one worker from beginning to end. In the team form the production process is carried out by a group of workers of various qualifications. Each member of the team fulfils the predetermined number of operations he is charged with; the team leader is responsible for over-all operations.

This latter form of labour organisation predominates in the production of ferroalloys. For example, an electric furnace is operated by a team of melters under the supervision of the chief melter. The latter is subordinated to the shift foreman and is responsible for the fulfilment of the task; the chief melter, besides being a team leader, participates directly in the production work.

The main task of proper labour organisation is to give maximum production with minimum labour and material costs. Industrial innovators achieve high efficiency through continuous improvement of technology and perfection of working operations.

The chief melter (who is at the head of a team operating a ferroalloy furnace) is in charge of the ferroalloy technological smelting process, takes part in furnace operations, in hot and cold repairs, and keeps a log of furnace operations.

The chief melter should know how to smelt different grades of ferroalloys by various methods and in various types of furnaces, basic electrical engineering and electrometallurgy, operation regulations and structural features of furnaces, and the physico-chemical properties of the materials used in the ferroalloy smelting processes.

He is responsible for the run of the smelting process in accordance with the approved technological instructions and time-table, for the functioning of the furnace and furnace equipment, for electrode operation and labour organisation within the team.

A melter should have the same qualifications as the chief melter, replace him in his absence and take part in all furnace operations.

He is responsible for the operation of the furnace-top and the functioning of furnace equipment.

The melter's mates prepare the necessary materials and tools and take part, on the chief melter's instructions, in all furnace operations and keep their working places in order.

They must know how to smelt various grades of ferroalloys, the basic principles of electrical engineering and electrometallurgy, operation regulations and structural features of furnaces, the physico-chemical properties of the materials used for the manufacture of ferroalloys.

The melter's mates are responsible for timely mixture charging and furnace operation, functioning of furnace equipment, cleanliness and order of their working places.

The forehearth operator taps metal and slag, carries out maintenance of the tap, inspects the condition of the furnace lining, repairs the arch, sets the furnace pouring devices and makes sure they are intact.

The forehearth operator must know the fundamentals of the technological process, the construction of ferroalloy furnaces, the physical and chemical properties of the materials used in furnace forehearth operations and the conditions for the solidification of metal and slag.

He bears responsibility for the tapping and pouring of metal according to schedule, for the functioning of the taphole, pouring devices and moulds, for the operation of the electrical tapper and auxiliary equipment (ladles, carriages, moulds, etc.).

All the members of the crew and, in the first place, the chief melter, should be highly qualified to ensure technically sound operation of complex electrometallurgical devices. In this connection, utmost importance is attached to the organisation of systematic technical training of production personnel, and to the improvement of their professional and technical qualifications.

New specialists are trained outside the plants by factory-and-workshop and vocational schools, as well as in the plants themselves, at technical courses, schools for foremen, etc.

One effective method of raising workers' qualifications is the so-called school of advanced working methods. Here, melters-innovators (or workers of other professions—forehearth operators, proportioners, etc.) teach their co-workers advanced, highly efficient work-

ing methods and show them rational ways and means of performing various operations, and teach them all that on the spot.

Scheduled operations, i.e., operations according to schedule constitute the most effective system of labour organisation in a ferroalloy shop.

Operation according to schedule means fulfilment of technological operations in strict sequence and within the time limits ensuring full execution of a pre-set task (plan).

The basis of shop planning is the output schedule for finished products. The schedule indicates the sequence of shut-downs for repairs and their character. It regulates the work of shop auxiliary branchworks (mixture delivery, metal dressing, slag dumping, etc.).

The daily time-table of shop operations comes to the teams in the form of work programme for the given shift.

The main furnace operations are also strictly scheduled either by time (charging and tapping for ferrosilicon furnaces, metal recovery from slag pots, etc.) or by electric power input (melting of charge portions in the production of refined ferrochrome).

Shop operations according to a time-table envisage conditions ensuring highly productive processing and, at the same time, make it possible efficiently to intervene and eliminate disturbances due to organisational or technical causes.

Technical Rate-Setting

Technical rate-setting, i.e., setting up technically substantiated rates, permits sound production planning, promotes higher labour productivity and lowers operation costs.

The task of technical rate-setting is to set up technical equipment quota and work and time rates.

Technical equipment quota means maximum per shift productivity of the furnace, most favourable operating conditions and best possible production and labour organisation. Technical equipment quotas are taken as a basis for calculating shop or enterprise productivity.

Technical time and output rates constitute the labour productivity task given to workers; these rates serve as a basis for the computation of workers' wages.

A time rate is the amount of working time necessary for a worker or a group of workers (team of furnace melters) to produce a unit of production in conditions of proper labour and production organisation. The time rate is expressed in man-hours.

The output rate means the amount of units of production expressed in accepted measurement standards (for example, tons), set as a task to a worker or a group of workers to be produced in a unit of

working time (hour, shift). The output rate is set in each separate case in accordance (with a slight decrease) with the technical productivity quota for the equipment under consideration.

It may be calculated as a quotient of division of the set duration of the shift by the time rate:

$$O = \frac{T}{t_{\text{rate}}}$$

and, vice versa

$$\frac{T}{O} = t_{\text{rate}}$$

where

O —output rate per shift, in units of production (tons of metal);

t_{rate} —time rate per unit of production (1 ton of metal), hours;

T —duration of a shift, hours.

All technical equipment output and time rates are variable; they change with the introduction of new, more efficient technological processes, new equipment, tools and devices, with the growth of the technical and cultural level of workers, improvement of the socialist forms of labour and production organisation.

Technical rates should be progressive, i.e., they should reflect a productivity level exceeding the mean level for the given shop and approaching the best performance results. Therefore, technical rates are periodically revised to take account of all the changes in production organisation in the preceding period.

Work Rates and Wage System for Workers

The rate system, which permits organisation and planning of wages, consists of the following elements:

- 1) rates of wages determining workers' hourly wages;
- 2) relative scales of wages determining the correct relation of wages among various types of work; differentiation of wages, depending upon the workers' qualifications, is implemented by these relative scales of wages;
- 3) rate-qualification reference book which contains the description of all types of jobs and the requirements to be fulfilled by a worker of a given qualification.

Every worker is given a certain qualification rate, depending on his profession and skill. The classification of profession and qualification under a certain rate is done on the basis of a reference book which takes the following circumstances into account:

- a) complexity of the technological process and equipment operated;
- b) minimum technical knowledge and professional skill required of a worker to perform a job corresponding to the qualification rate;

c) responsibility for the work performed.

All processes connected with the operation of production branch-works of metallurgical shops in the Soviet Union are divided into ten qualification rates while auxiliary and repair operations are divided into eight qualification rates. The qualification rate is given to a worker by the shop superintendent after a corresponding technical examination and depends on his qualification and seniority.

Wage rates are set separately for each qualification rate: the minimum for the first qualification rate and highest for the most qualified.

Wage systems in metallurgical industry are divided into two main groups:

- a) according to the time spent on the job (time-wage system);
- b) according to the results of the job (piece system).

In the case of the time-wage system, the worker's wages are computed according to hours of work per day or days per month and according to his rate of wages. This system is used when the production output cannot be determined accurately, as well as for the wages of auxiliary and duty personnel.

A time-premium system of wages is used in metallurgical industry, when time workers are paid a bonus for the qualitative results of their work, provided the production plan of the department, shop and the equipment operated is fulfilled and overfulfilled; the bonus ranges from 15 to 20% of the monthly wage rate.

With the piece system, the worker's wages are calculated according to the amount of work performed, i.e., the wages are proportional to production output. This form of wage system stimulates self-improvement, as well as betterment of production methods. The basis for the wage calculation in this case is the piece rate per unit of production which is determined by the rate of wages and the fixed-time rate. The wages of a worker in the piece system are calculated by multiplying the piece rates by the amount (pieces or units) of manufactured products in a unit of time (shift, month).

The piece system is more progressive, for it combines the principle of personal interest and state interest.

"From each according to his ability, to each according to his work" is the principle of socialism. The piece rate best expresses this principle.

In the iron and steel industry the production workers of a metallurgical unit (furnace, mill, etc.) and related departments of metallurgical and coke-oven shops are paid according to the piece-premium wage system which provides piece wages and a bonus—20% of the monthly wage rate for the fulfilment of the monthly production plan and an additional 2% of the monthly pay for each per cent of the over-fulfilment of the plan.

Instead of a premium for the fulfilment and overfulfilment of quantitative tasks, a bonus for an increase in first-grade production output, for the fulfilment of time-tables, economy on materials, etc., is given the workers of production branchworks whenever an improvement of qualitative indices of the plan is necessary.

The total sum of premiums paid for economy on materials, fuel and electric power should not exceed 40% of the sum saved.

The workers of the main shops working on production units and in departments, where it is impossible to set task plans and premium incentive for the fulfilment and overfulfilment of plans, are paid according to a piece-progressive system of wages: output in excess of the monthly plan is paid from 1.5- to 2-fold of the ordinary piece rates.

Chapter 3

PRODUCTION COSTS

Basic Principles of Ferroalloy Economics

Production of ferroalloys^{*} is related, qualitatively and quantitatively, to steel production. The larger the output of steel and the bigger the output of alloy steels, the bigger the need in ferroalloys.

In turn, the level of steel output determines the development of all branches of the national economy; therefore, the production of ferroalloys is of utmost importance for the national economy.

The production of ferroalloys in the Soviet Union is constantly on the up-grade. The growth in the production of ferroalloys smelted in electric furnaces, as well as by other methods, is shown in Table 69 (in per cent of the 1946 level).

Table 69

**Dynamics of the Growth of Ferroalloy Output in the U.S.S.R.,
in % of the 1946 level**

Ferroalloys	1950	1955
Total	154.1	295
Including:		
ferrosilicon	143.2	279
ferrochrome	156.7	375
ferrotungsten	210	506.6
ferromolybdenum	100.9	286
ferrovandium	263	364
ferrotitanium	738	2,210
Total of electric furnace manganese alloys	223.6	348
Including:		
metallic manganese	587	214
silicomanganese	167	437

The development of ferroalloy industry has somewhat outpaced the growth of steel production.

The output of ferroalloys smelted in electric furnaces is determined by the possible productivity of single furnaces which, in turn, is calculated by the following formula:

$$Q = \frac{W \times \cos \varphi \times K \times 24}{A} N,$$

where

W is transformer power rating at the working voltage, kVA;

$\cos \varphi$ —power factor;

K —coefficient of furnace capacity use;

N —number of days of operation in the time period considered;

A —specific electric power consumption, kWh/ton.

For larger furnaces, operating continuously, K is approximately equal to 0.97-0.98. The loss of 2-3% of power rating is due to line voltage fluctuations, insufficient sensitivity of automatic regulators, as well as due to a certain drop of power input in the electrode repositioning operation (furnaces with electrode holders not equipped with clamps).

For electric furnaces of 2,500-3,500 kVA rating, operating with periodic melting of the charge, K is considerably lower, as power input losses are unavoidable as the result of the following causes:

- 1) decrease of power input and interruption of power supply during the tapping of metal and slag;
- 2) unstable furnace operation with open electric arcs;
- 3) decrease in power input during the charging periods and power input raise;
- 4) interruption of power supply for the repositioning of electrodes.

Because of that factor K varies within the limits of 0.75-0.85, depending on furnace design and technology features (frequency of tappings of metal and slag, grade of metal smelted, etc.). The task of the Soviet ferroalloy melters is to find ways of cutting down all idling periods, power input losses and of raising furnace productivity.

The number of nominal days of operation N of a furnace is given as the difference between the number of calendar days during the period under consideration and the number of down-time days for overhaul and maintenance repairs.

Specific power consumption A for the smelting of an alloy is set annually for each plant, depending on local conditions (furnace type and electric circuitry, quality of raw materials, etc.).

Estimation of the requirements in ferroalloys for long-term planning is a difficult task, and it is solved on the basis of the data on the specific consumption of ferroalloys per ton of steel produced in the previous years, with due consideration of changes in alloy steel output for the planned period.

The needed amount of deoxidisers is determined by the level of steel output and the ratio between the amounts of the rimming and

killed steels produced. According to E. M. Alexeyev, since 50% of all steel produced is accounted for by killed steels, the consumption of deoxidisers per ton for the entire amount of steel produced may be assumed to be as follows: electrothermic ferrosilicon (in terms of 45% ferrosilicon)—7.5 kg, blast-furnace ferrosilicon (10% Si)—9 kg, and carbonaceous ferromanganese—7.8 kg.

The requirements in ferroalloys, used for purposes other than steel production (ferrotitanium, ferroboron, crystalline silicon, etc.) is determined by direct calculation.

Cost of Ferroalloys

Production costs are expenses incurred in money for the manufacture and sale of products.

Systematic cuts in production costs in the U.S.S.R. are of paramount importance, since it is a source of accumulation in national economy, as well as of improvement in the welfare and cultural standards of the Soviet people.

The decrease in production costs is an important task of the state plan.

Ferroalloy production costs are made up of the following main expenses:

- 1) cost of raw and basic materials;
- 2) cost of electric power;
- 3) cost of electrodes;
- 4) cost of processing; this item includes the cost of all auxiliary materials used (refractories, sand, clay, etc.) workers' and engineers' wages, cooling-system expenses, steam, compressed air, etc., wear and tear, services by auxiliary shops (mechanical, transport, electric, etc.);
- 5) cost of packing;
- 6) overhead expenses.

The distribution of costs of some ferroalloys, given in Table 70, shows the direction in which an effort should be made to bring down the cost of each alloy.

In smelting ferrosilicon, silicon calcium and crystalline silicon particular attention should be paid to reducing electric power consumption, the cost of which takes up to 50% of total alloy costs.

Alloy costs may be cut down tangibly by reducing consumption of raw materials, and that despite the fact that the efficiency of the use of silicon (90%) and coke carbon is high and the turnings are used practically completely. This may be attained by decreasing the losses incurred in the pre-smelting preparation of raw materials, which are very high at present (25% in quartzite and 18% in fine coke).

Table 70

Distribution of Costs of Some Ferroalloys

Cost Item	Expenses, % of full prime cost										
	4.5% fer- rosilicon	7.5% fer- rosilicon	crystalline silicon	silicomane- gium	carbonace- ous ferro- chrome	medium- carbon ferro- chrome	extra-low carbon ferro- chrome	ferrotita- nium	aluminu- m thermo- mic ferro- chrome	ferrotung- sten	ferromo- lybdenum
1. Raw and basic materials . . .	29	21	23	28	38	67	64	79	76	97.9	98.14
2. Electric and power costs . .	33	37	16	29	28	8	8	1.0	—	0.36	0.16
3. Electrodes . . .	2	3	6	4	3	~ 1	2	—	—	0.14	0.04
4. Processing and overhead costs*	36	39	55	39	31	24	26	20	24	1.6	1.66
Total	100	100	100	100	100	100	100	100	100	100	100

* —distribution of costs in the last stage of processing only

The use of closed-top rotating-bath furnaces provides the possibility to economise (according to available data) up to 10% on raw materials (as compared to manufacture in open-top furnaces) by decreasing silicon losses and reducer evaporation. The use of the sensible heat of gases gives a saving of up to 40% of costs of the fuel consumed. Moreover, expenses on wages are brought down in the use of closed-top furnaces by full mechanisation of the charging operation.

The main task in the production of such alloys as ferrochrome and ferromanganese is to save raw materials, chiefly silicon chrome and silicomanganese. Full recovery of metallic prills should be organised; this will raise considerably the recovery of the basic element, i.e., to lower raw-material and electric-power consumption. An important task is utilisation of slags, which are a precious building material.

As the result of the development and realisation of the slag process in silicon-chrome manufacture the costs of raw materials, electric power and wage expenses may be brought down.

Economy of raw materials and maximum recovery of the prime element are of major importance for the production of ferrotungsten, ferromolybdenum, ferrotitanium and other alloys, an effort to save all other items should be striven for too.

Of considerable importance are the increase in plant capacity and productivity of single production units. Initial and operational expenses in auxiliary shops, considered as units of plant productivity, as well as administrative expenses, are reduced along with an increase in plant capacity.

Initial costs per unit of output and specific power consumption are diminished, while the labour productivity of furnace operators and furnace yield per unit of power rating increase with the rise in electric furnace rating.

Table 71 lists the personnel of a six-furnace smelter shop, equipped with mechanised charging, alloy teeming by pouring machines and automated mixture conveyance system. It should be emphasised that the number of workers employed in a shop with 10,500-kVA furnaces is the same as in that with 16,500-kVA furnaces, although productivity per worker in the latter case is 1.5 times as high and wage expenses are correspondingly 1.5 times as low.

Material handling operations should be mechanised to the maximum, as approximately 20% of workers are engaged at present in these operations. The reduction of this expenditure item through mechanisation is an important source of reducing ferroalloy costs. Mixture preparation and proportioning, as well as the dressing of finished products, should also be mechanised and automated.

Improvement in the pre-smelting preparation of raw materials may considerably enhance labour productivity and decrease ferroalloy costs.

Drying of fine coke will make for greater precision of proportioning and ensure a stable and uniform run of furnaces. Crushing, sizing and washing of quartzite directly on the mining sites eliminate useless transportation of fines (which are dumped), whose amount may reach as much as 25% of the weight of the quartzite shipped to the plant.

Table 71

**Approximate Number of Workers, Engineers and Employees
in Six-Furnace Ferrosilicon Smelter Shop with Mechanised Charging**

Position	Number of workers		
	In four shifts	substitutes for those on leave	total
<i>Workers</i>			
Senior melters	24	2	26
Melter-charging machine operators	48	5	53
Hearth operators	48	5	53
Crane operators	28	3	31
Electrode operators	4	—	4
Mixture makers and proportioners	12	1	13
Finished-products warehouse workers	12	1	13
Shift electricians	12	1	13

Position	Number of workers		
	in four shifts	substitutes for those on leave	total
Maintenance electricians	8	—	8
Shift furnace and auxiliary equipment fitters	9	1	10
Maintenance fitters	10	—	10
Masons for refractory brick work	4	1	5
Utility workers	3	1	7
Total	—	—	246

Engineering Staff

Shop superintendent	—	—	1
Deputy shop superintendent (chief foreman)	—	—	1
Shift chiefs	4	1	5
Shift foremen	4	1	5
Chief shop electrical engineer	—	—	1
Electrician-foreman	—	—	1
Chief shop mechanical engineer	—	—	1
Fitters' foreman	—	—	2
Mixture foreman	—	—	1
Rate setter	—	—	1
Planning engineer	—	—	1
Total	—	—	20

Employees

Shop office head clerk	—	—	1
Accountant	—	—	1
Time-keeper	—	—	1
Store-keeper	—	—	1
Office cleaners	8	1	9
Preparation and distribution of aerated water	4	1	5
Total	—	—	18
Total shop staff	—	—	284

Mixing (so as to obtain a homogeneous composition) and drying of the chrome and manganese ores used for smelting carbonaceous and foundry ferrochrome, carbonaceous ferromanganese and silico-manganese will ensure mixture composition uniformity, which is very important for the improvement of over-all operating results.

One of the main factors determining the cost of ferroalloys is how far the plant is situated from the sources of power, raw materials and from the consumers. The decisive point in the choice of the plant's site is the amount of raw materials and fuel handled per unit of production, including the shipping of finished products to consumers.

According to E. M. Alexeyev, it is most advantageous to build power-consuming ferroalloy plants in the immediate vicinity of hydropower stations or coal mines.

This is corroborated by the table given below, which compares the amounts of raw materials consumed per ton of some grades of ferroalloys and the amounts of fuel needed to generate power essential for smelting one ton of alloy.

Table 72

**Raw-Material and Coal Consumption for the Manufacture
of One Ton of Ferroalloys, Tons**

Materials	Ferrosilicon		Refined ferrochrome	Carbaceous ferromanganese	Silicomanganese
	Cu75	Cu75			
Basic raw materials	2.25	3.05	5.7	3.2	3.0
Coal with calorific power of 5,000 kcal/kg for power generation	4.3	7.7	6.8	3.2	3.7

As the table shows, ferromanganese- and silicomanganese-smelting plants may be situated both in the vicinity of power stations and near raw-material sources.

Considering the geographical distribution of consumers, it may be more advantageous to build plants at equal distance from the raw-material and power sources, but nearer to the consumers.

The ways of reducing costs we have just considered do not include all the possibilities available. For each plant, for each particular situation, new problems of cost reduction arise, and their solution is of utmost importance to the enterprise.

Chapter 4

BASIC PRINCIPLES OF SAFETY ENGINEERING AND INDUSTRIAL SANITATION

Industrial work in a socialist enterprise must be safe and non-exhausting, and it is so. At the same time, maximum labour productivity should be achieved, for V. I. Lenin said, "In the final count, it is labour productivity that is the most important, the prime factor for the victory of the new social system."

The goal of safety engineering and industrial sanitation is to prevent accidents and to create favourable and safe working conditions in the shop and at the working places.

Industrial injuries are caused mainly by violation of technical and routine regulations, as well as by non-observance of the rules of workers' behaviour. For example, operation of a technically faulty machine or improper planning of operations and non-coordination of work performed by workers may lead to an accident. Very often accidents are caused by the infringement of rules of workers' behaviour, for example, improper use of protective clothing, performance of work that has not been ordered, etc.

Violation of safety regulations, therefore, entails punishment according to routine regulations; when the infringement of regulations causes an accident injurious to other workers, those guilty may be brought to trial in particularly severe cases.

Every newly employed worker is briefed by the safety engineering department and is then given a thorough illustrative briefing on operating procedures as well as on safety regulations by the foreman or the shift foreman directly at the working place. It is forbidden to start work without such instructions. The men responsible for instructions and the compilation of regulations are the foreman and the shop superintendent. If transferred to some other job, even for a short time, the worker should ask to be instructed at the new working place and go to work only after that. Those working with mechanisms, electrically-operated appliances or melted metal should possess the required technical knowledge; to this end they must take a theoretical course not later than three months after joining the enterprise and pass the qualification exams.

Administrative personnel should explain to workers why it is necessary to observe safety and industrial sanitation regulations and impose severe penalties on those guilty of infringements.

Electrothermic Ferroalloy Shops

Shop Site and Layout

Ferroalloy shops should be built on the territory of the plant in such a way as not to disturb the flow of materials and to preclude opposing traffic and intersections.

Adequate aeration requires that the longitudinal axes of newly erected smelters should be perpendicular or at an angle of minimum 30° to the direction of the prevailing winds in the locality. The direction of the wind should be from the working area to the smelter aisle.

Railway tracks should have a minimum of curves, crossings and inclines.

All newly erected ferroalloy works should be equipped with special mechanised pile-driving, slag and dressing departments, as well as storages for the finished products.

The location of these departments should be selected with a view to minimising haulages and streamlining production.

Roads with pavements are to be constructed between production shops.

Stock Yard and Mixture Conveyance

The building of the stock yard should be fire-resisting, roofed and equipped with railway tracks along its longitudinal axis.

The columns of the stock yard should be concrete or metal.

Crushing or mechanical equipment, as well as conveyors, should be situated in the stock yard with due regard to the following:

- a) safety and accessibility;
- b) personnel safety when grab-bucket cranes move over the working places;
- c) layout of equipment; it should not impede the unloading of railway cars and transportation of materials within the stock yard;
- d) crushing devices, producing a lot of dust, should be hermetically sealed and provided with exhaust ventilation;
- e) all rotating assemblies of drives should be fenced off: platforms and stairs by metre-high hand-rails; coamings or "toeboards", 100 to 200 mm high, should be placed along the edges of the platform at the foot of the railing.

Special gangways should be provided over the crossings of conveyors, bins, etc.

Railway tracks should not be obstructed within the limits of clearances (2.5 m from the axis of the track), and should be equipped with a flooring between the rails.

Natural ventilation (skylights, fanlights, etc.) as well as lighting according to code specifications, should be provided for in the raw-materials building.

The incline of belt conveyors without partitions should not exceed 25° , and 50° in the case of conveyors with partitions.

Materials stuck in the bins should be poked from above with long pipes. Poking when there are men inside the bin is permitted only in exceptional cases, and then special precautionary measures must be taken.

The stock yard should be constantly kept in good order.

Proportioning Department

The building accommodating the receiving hopper of the inclined conveyor should be well-ventilated and spacious to facilitate repairs and servicing.

A distributing station communicating with the stock yard by an alarm signal system should be provided for distributing the mixture to the suspended bins.

All rotating parts of drives should be fenced off.

Building of the proportioning branchworks should be provided with facilities for natural air infiltration and exhaust ventilation.

Proportioning branchworks area should be isolated from the smelting aisle by a continuous wall in order to prevent gas infiltration.

The movement of cars filled with mixture materials is controlled by push-button devices; their dumping should be mechanised.

These regulations apply to the proportioning branchworks forming part of the stock yard.

Smelter Shop Buildings

The walls of a smelter shop should be made of refractory materials.

All metallic structures should be painted.

The roofs are designed and erected to carry additional dust loads; provisions should be made for their rapid and convenient cleaning.

All concrete columns should be enclosed in metal to a height of 1.5-2 m from the shop floor.

Flooring in the smelter shops is made of cast-iron ribbed plates.

The working platform should be at least 15 m wide in the case of smaller furnaces and 20 m wide in the case of larger ones.

The working floors should be fenced off by handrails 1.2 m high

and sheeted all around at the bottom to a height of 200 mm. The stairs should be minimum 1 m wide and the tread should be at least 270 mm wide. The building should have aeration facilities; fresh air from plenum ventilation devices should be conveyed directly to the furnace.

Installation of air-fanning devices is imperative.

The furnaces should be screened to protect operators from heat radiation.

Furnace Layout and Operation

Melting furnaces in newly erected shops should be arranged in a line along the length of the building, with the intervals between their axes being not less than 18 m for 3,500-kVA furnaces, 30 m for open-top furnaces of 10,500-16,500 kVA and 24 m for closed-top furnaces of the same rating.

Furnace low-voltage circuits should be railed off to preclude all possibility of operating personnel coming into contact with them. The furnace should be equipped with the necessary metering and signalling devices; without them furnace operation is prohibited. The furnace shell should be well grounded. The exhaust pipes and gas-suction appliances should be kept in condition to evacuate completely the gases evolving in the furnace.

Particular attention should be paid to the system of oxygen supply for taphole lancing and the condition of hoses and valves. The hoses and the entire system should be effectively protected from oil, since its interaction with oxygen causes explosions.

Floors of the working and electrode decks should be made of non-conducting material.

Shunt circuit breakers should be well grounded and provided with an isolated stand and railings.

Only well-instructed operators are allowed to joint electrode shells.

The furnace can be switched on only by the senior melter with the assistance of a regulating operator and a shift electrician and in the presence of the shift foreman.

Hoists are operated in accordance with the regulations set for hoisting mechanisms.

Furnace-charging should be mechanised to the maximum.

The forehearth platform should always be dry.

The melter should work only in protective clothing:

- a) the cloth trousers and jacket should not be tucked in. This allows metal or slag splashes to roll down onto the floor;
- b) brim of the hat should be lowered; blue spectacles should be fixed to the hat to protect the eyes from electric arc or melted metal radiation.

When an iron rod is used for poking the charge it should be grounded through contact with the furnace shell or a roller.

Only dry materials should be charged; moist ones may cause eruptions of red-hot particles.

It is categorically prohibited to operate a furnace if there is a water leakage. The furnace should be shut down immediately for necessary repairs.

It is forbidden to do any work with ladles or slag pots installed under the taphole. Furnace tapholes can be cleaned and dressed only from a special platform. It is prohibited to stand directly in front of the taphole.

It is prohibited to do any work under the unrailed sections of the low-voltage circuit when the power is on or to switch on or off the shunt circuit breakers in such conditions.

Workers should see to it that their instruments are in order. It is forbidden to work with faulty or inappropriate tools. All instruments used for metal or slag should first be dried.

Preparation of Ladles, Moulds and Slag Pots, and Teeming of Metal

The taphole should be situated at a convenient height for the operator; if necessary, special platforms or stands should be provided.

To prevent damage to lining in the taphole zone it is necessary to check it systematically; if there are any defects, the taphole should be immediately repaired.

People not involved in the operation should not be allowed near the furnace during the tapping process.

Ladles should be filled with the melt to a height of 100-200 mm below the brim.

All receptacles prepared for tappings should be checked by fitters. Special trestles—working benches—should be provided for cleaning ladles.

A newly lined or repaired receptacle should be thoroughly dried.

Receptacles should be wheeled in and out from under the taphole only by mechanisms.

Tracks intended for transportation of liquid metal and slag should be regularly checked and the flaws immediately eliminated.

Pouring receptacles installed on carriages should be stable. If the ladle carriage makes this impossible, it should be immediately replaced with a faultless one.

Before wheeling out a receptacle, one must be sure that there are no men and objects on the tracks and only then can power be applied to the winch or command given to the crane operator.

The procedure for lifting a ladle or a slag pot full of liquid metal or slag is as follows: when the ladle has been lifted to a height of 100 mm a "stop" signal is given, and it is only when one is certain of proper connections and reliability of crane brakes, that a "go" signal is given again.

Liquid metal or slag should be conveyed at low speed, care being taken to avoid swinging the load.

The alloy is poured with the aid of auxiliary hoists (Fig. 88), smoothly, without jerks.

Following the pouring of ferrosilicon the ladle is returned to the vertical position and sent to be cleaned. In the case of ferrochrome teeming the dumping of the slag scull is permitted only an hour later.

It should be borne in mind, when stripping alloy ingots from the moulds, that ferrosilicon ingots may break in the process of transportation, while ingots of refined ferrochrome, and especially those of medium-carbon grade, may explode when cooled. Therefore, ferrochrome ingots should be stripped either immediately after the pouring or after the final cooling. When ferrosilicon ingots are transported, workers should not stand closer than 5 m from the suspended ingot.



Fig. 88. Teeming of ferrosilicon

Granulation of Ferroalloys

Granulation tanks should be approximately 6 m deep, its edges about 600 mm above the floor level. The water level in the tank should be 300 mm below its upper edge.

Granulation tanks should be cleaned periodically; the temperature of the water in the tanks should not exceed 70°C.

To avoid local spots of violent evaporation the metal jet should be disintegrated by a jet of water into granules of a predetermined size.

Water conveyed to the nozzle of a sprayer should be under a pressure of not lower than 3 kg/cm².

The dimensions of the receiver eye for foundry ferrochrome and 30% silicon chrome should be 70×70 mm, while that for the 75% ferrosilicon and 50% silicon chrome—40×65 mm.

Persons not concerned with the operation should not be allowed to come near the granulation installation.

Granulation tanks should be equipped with a steam exhaust system to ensure adequate visibility in winter.

Settlers are to be provided to avoid the leakage of granules or slag into sewers; they have to be cleaned periodically.

Unloading of metal from the granulation tank should be mechanised.

Slag Yard

A special slag yard for the cooling and shipping of slags should be included in all newly erected ferroalloy smelters and equipped with electric overhead cranes.

A slag yard should be paved with ribbed cast-iron plates.

The slag is loaded into special cars, preferably of the drum type.

When a slag yard is equipped with facilities for recovering prills from disintegrating slags, the entire system of conveying slag to the vibrating screens, as well as the screens themselves, should be hermetically sealed and vacuumed.

Dressing and Storing of Finished Products

The premises for cooling, dressing, packing and storing of finished products should be built of fire-resisting materials and may be located apart from the smelter shop.

Pile drivers are installed at one end of the building, at a distance of at least 10 m from the area intended for metal dressing.

All rotating mechanisms in the winch room should be adequately railed off, while current-carrying parts should be insulated.

A finished-products storehouse should be provided with sufficient floor space to ensure safety of operations. Avoid any obstructions in a storehouse area.

Furnace Overhaul

"Hot" furnace repairs can be effected only when the power is disconnected. Water-cooled screening installations are used for protection against the effect of heat radiation.

The blasting of salamanders is permitted in exceptional cases in "cold" furnace overhauls, provided, of course, the existing regulations are observed. Ammonal alone should be used in blasting operations; other explosives are prohibited.

In dismantling the lining, the furnace should be cooled with water not before 2-4 hours have elapsed since its shut-down; otherwise blowouts may occur.

The shop office should appoint a person responsible for the execution of the overhaul.

Exothermic Smelter Shops

An exothermic smelter shop consists usually of the following branchworks:

- 1) stock yard, which is equipped with installations for the pre-smelting preparation of mixture materials (crushing, drying);
- 2) roasting;
- 3) proportioning and mixing;
- 4) smelting;
- 5) cooling and dressing;
- 6) finished-products storehouse.

The mixture materials used in exothermic smelting should all be milled; the milling, proportioning and mixing processes generate dust. Therefore, particular emphasis should be placed on dust exhaust from points where it is generated; processes causing abundant evolution of dust should be conducted in hermetically sealed chambers.

It is recommended to carry out exothermic smelting in a separate priming chamber so as to preclude the infiltration of gaseous products into the ambient atmosphere.

Joint calcination and drying of saltpetre with ferrosilicon, electron or aluminium turnings are prohibited. The maximum permissible temperature for drying saltpetre is 150°C, while that for electron turnings is 300°C.

Minimum dust contamination should be ensured in the roaster works when roasting raw concentrates; special precautionary measures should be taken to prevent the ignition of oily matter contained in the concentrate when it is charged.

Only dry materials should be proportioned.

When smelting processes are under way, admission to the priming chamber is forbidden; no foreign matter should be stored in this chamber.

If the smelting process is effected with bottom priming, there should not be more than one charge batch placed on the hearth prior to priming.

The smelting shaft should be thoroughly dried before the smelting process.

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